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FOREWORD

The Coal Division of the Institute was organized in 1930, and the first volume embracing exclusively coal papers was published in that year, under the chairmanship of Howard N. Eavenson, who was the Chairman in 1931 also and is now President of the American Institute of Mining and Metallurgical Engineers. The Division Chairman in 1932 was Mr. Thomas G. Fear. Volumes were issued by the Coal Division in 1931 and 1932 but none in 1933. The present volume includes papers that were presented at the joint meeting of the Coal Division and the Anthracite Section held at Hazleton, Pa., Oct. 14 and 15, 1932, and at the joint meeting of the Division and the Ohio Section of the Institute held at Columbus, Ohio, on Oct. 27 and 28, 1933, as well as papers presented at the Annual Meetings of the Institute and the Division in New York in 1933 and 1934.

The studies that have been in preparation for some time pertaining to the Classification of Coal, embraced in papers published in the Coal Division volume in 1930 and in this volume, have been of service to the coal industry during the past year. The preparation and inauguration of Codes governing the bituminous mining industry under the N.I.R.A., and the negotiations looking towards the preparation of the Code governing the anthracite mining industry, found it desirable to establish commercial classifications of grades or ranks of solid fuel. The studies embracing classification included in the volumes above mentioned were of service. It is unnecessary to refer in detail to the various papers included in the present volume, because they speak for themselves.

It may not be out of place for me, after many years connection with the production, preparation and marketing of both anthracite and bituminous coal, to include some observations based upon that long period of experience. In common with all other industries in this country and in the rest of the civilized world, coal has not escaped the devastating effects of the depression, from which it appears we are now slowly emerging. Coal has not only been affected by the general "slump" in industrial activities, but its field of usefulness has been seriously invaded by other fuels, such as crude oil and natural gas. The efforts on the part of coal producers to keep alive against the competitive fuels, plus the depression, have resulted in commendable improvements in mining and preparation facilities and activities.

With the adoption of the Codes in bituminous mining, substantial progress has been made in the direction of sensible cooperation between producers and the different regions and different grades of coal reaching the market. This cooperation has also been legalized by the decision of the United States Supreme Court in what is known as the Appalachian Coals Case. The benefits mentioned have been and are still substantial, but the greatest benefit that coal has been blessed with was a real "honest to goodness" old-fashioned winter.

It is a dangerous thing to make prophecies, but I feel confident that, with sensible cooperation and continued adoption of proven improvements in mechanical apparatus, and particularly fair and all-embracing understanding between producers and employees, the solid-fuel industry may look forward to continued sharing in the rising tide of returning prosperity.

Eli T. Conner,
Chairman, Coal Division, 1934

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Occurrence and Flow of Gas in Pocahontas No. 4 Coal Bed in Southern West Virginia

By Charles E. Lawall* and Lee M. Morris,† Morgantown, W. Va.

(New York Meeting, February, 1933)

This paper represents the results of a study to determine the source, mode of occurrence and conditions influencing the flow and liberation of large volumes of inflammable gas in the Pocahontas No. 4 coal bed in southern West Virginia. A study of this kind is of interest in connection with mine ventilation, because in the removal of gases from coal mines it is very important to know the origin, the occurrence and laws of gas flow. The study was started in April, 1931, and the data and observations were obtained throughout a period of 15 months of field work.

Geology

The country where this study was made is mountainous and rugged; great cliffs, precipitous slopes and deep narrow gorges are found throughout the area. The outcropping rocks consist of stratified layers of sandstone, shale, limestone and deposits of coal of Carboniferous age. The Pottsville series of the Pennsylvanian, lying at the base of this period, cover the greater portion of the area. A stratigraphic section of the Pocahontas group of the Pottsville series is shown in Fig. 3. The older rocks outcrop in the extreme southeastern part of the area, but are under drainage in the central and northern parts due to the gradual northwest dip of the strata.

The structure has a general northeast-southwest trend (Fig. 1). North of Dry Fork anticline the strata have been only slightly disturbed by movements of the earth's crust; it is now a long monocline. Southeast[1] of Dry Fork anticline the strata are warped into great arches and deep troughs and are broken by overthrust faults. All of this folding is probably the result of the intense folding associated with the formation of the Appalachian Mountains.

Pocahontas No. 4 Coal Bed

The Pocahontas No. 4 bed, lying approximately 550 ft. above the base of the Pottsville series, is one of the thickest and most gassy coal beds in

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Manuscript received at the office of the Institute Jan. 20, 1933.

* Director, School of Mines, West Virginia University.

† Assistant Geologist, West Virginia Geological Survey.

[1] Virginia Geological Survey Reports.

southern West Virginia. This bed is a low-volatile, high-carbon coal and in general has no definite joint system. The general section of the coal is shown in Fig. 2. A structural map based on this coal bed is shown in Fig. 1 with its approximate minable area shown in Fig. 3. It is

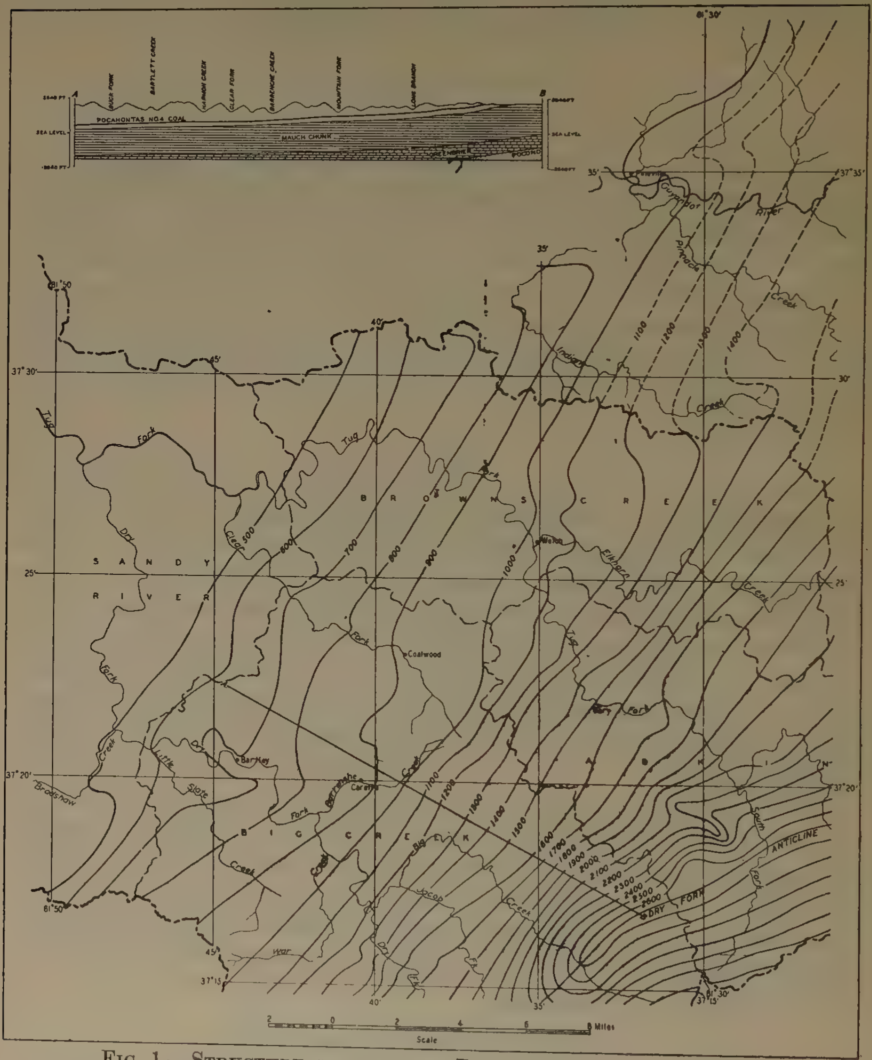

Fig. 1.—Structure contours on Pocahontas No. 4 coal bed.

especially gassy where the bed lies below water level and is under thick cover. Along its outcrop where the cover is thin and the bed is exposed, comparatively little gas is encountered in mining. Two mines working this bed liberate daily approximately 13 million cubic feet of inflammable gas. If this gas could be conserved its value, based on a price of 10 c. per

1000 cu. ft., would amount to $1300 per day. Data were gathered on six mines that liberate approximately 23 million cubic feet of inflammable gas every 24 hr. during normal production of coal. The probable value of so much gas passing out of the mine with the ventilating current would be $2300 per day.

In all of these gassy mines, the largest volume of methane is liberated from sections that are advancing into virgin coal. This condition often necessitates an interruption of mining until some of the gas bleeds off, which indicates that liberations of inflammable gas vary largely with the rate at which the coal is broken up in mining.

UNDERGROUND EXPERIMENTS

All the underground experiments and observations were made in a mine designated as mine A in this paper (Fig. 4). This mine is in the Pocahontas No. 4 coal bed, which at this place averages 6 ft. 2 in. in thickness and dips N. 45° W. 2 per cent. The superincumbent strata are from 560 to 1400 ft. thick. The experiments are divided into two groups with objectives as follows:

1. To locate the source of the gas.
2. To measure the flow and pressure of the gas.

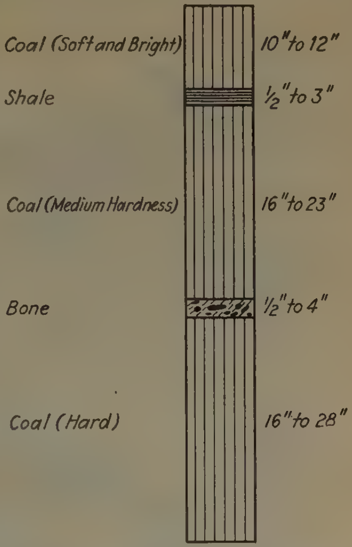

FIG. 2.—GENERAL SECTION OF POCAHONTAS NO. 4 COAL BED IN SOUTHWESTERN McDOWELL COUNTY.

The location of each station is shown in Fig. 4. Each station was selected so that various sections of the mine would be included in the study. This was done to take into account such factors as the rate of mining; length of time coal had been exposed; proximity of surrounding workings and solid coal; variations due to the presence of water and changes in the physical character of the coal.

EXPERIMENTS TO LOCATE THE SOURCE OF THE GAS

Station No. 1.—The location of experimental station No. 1 is shown on Fig. 4 (at the upper right corner).

A hole 21 ft. deep was drilled horizontally into the center of the coal bed; water was struck at a depth of 12 ft. The hole was cased to a depth of 5 ft. by inserting a 1½-in. iron pipe into the hole and packing oakum around it. Pressure and flow readings were then taken, and the hole left open for the gas to drain.

A hole 7 ft. deep was drilled into the roof strata at an angle of approximately 45°. The hole was cased to a depth of 5 ft. by the insertion of a 1¼-in. iron pipe with oakum packed around it. There was no flow of gas and the hole was left closed to determine whether gas would accumulate in it.

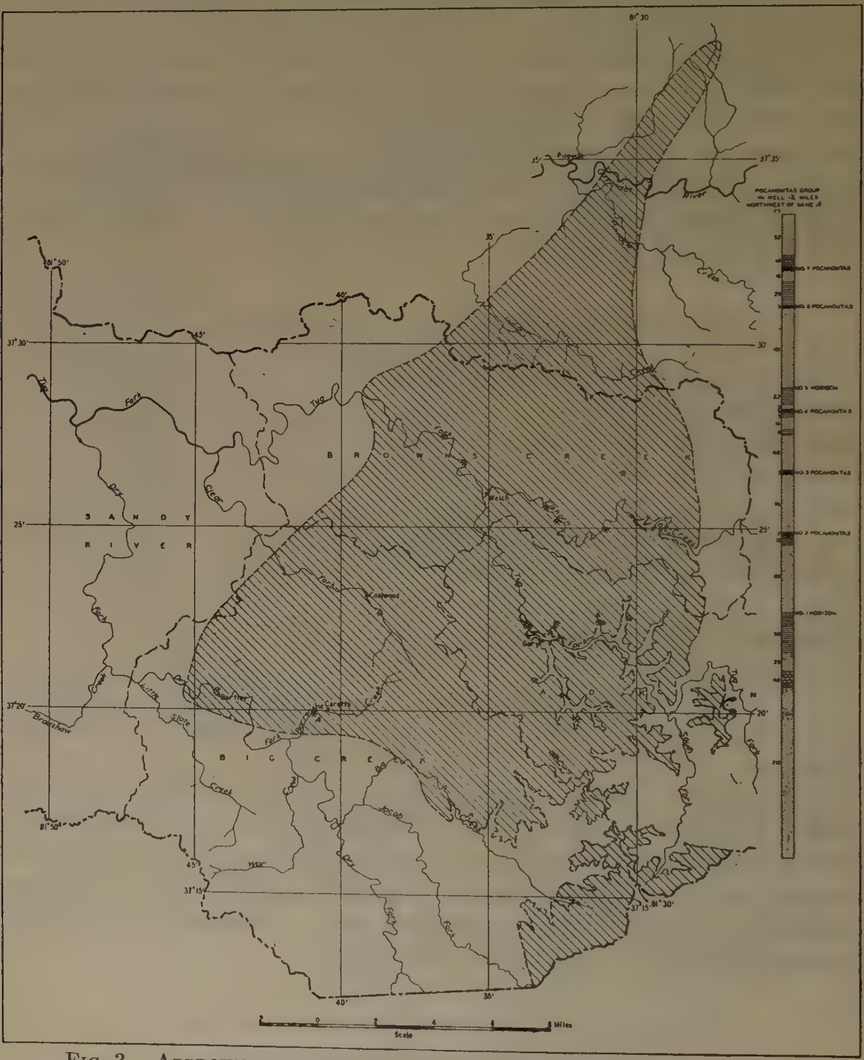

Fig. 3.—Approximate minable area of Pocahontas No. 4 coal bed.

A hole 8 ft. deep was drilled into the floor at an angle of approximately 50° with the horizontal. The hole was cased to a depth of 5 ft. by inserting a 1¼-in. iron pipe and packing oakum around it. There was no flow of gas and the hole was left closed to see whether gas would accumulate.

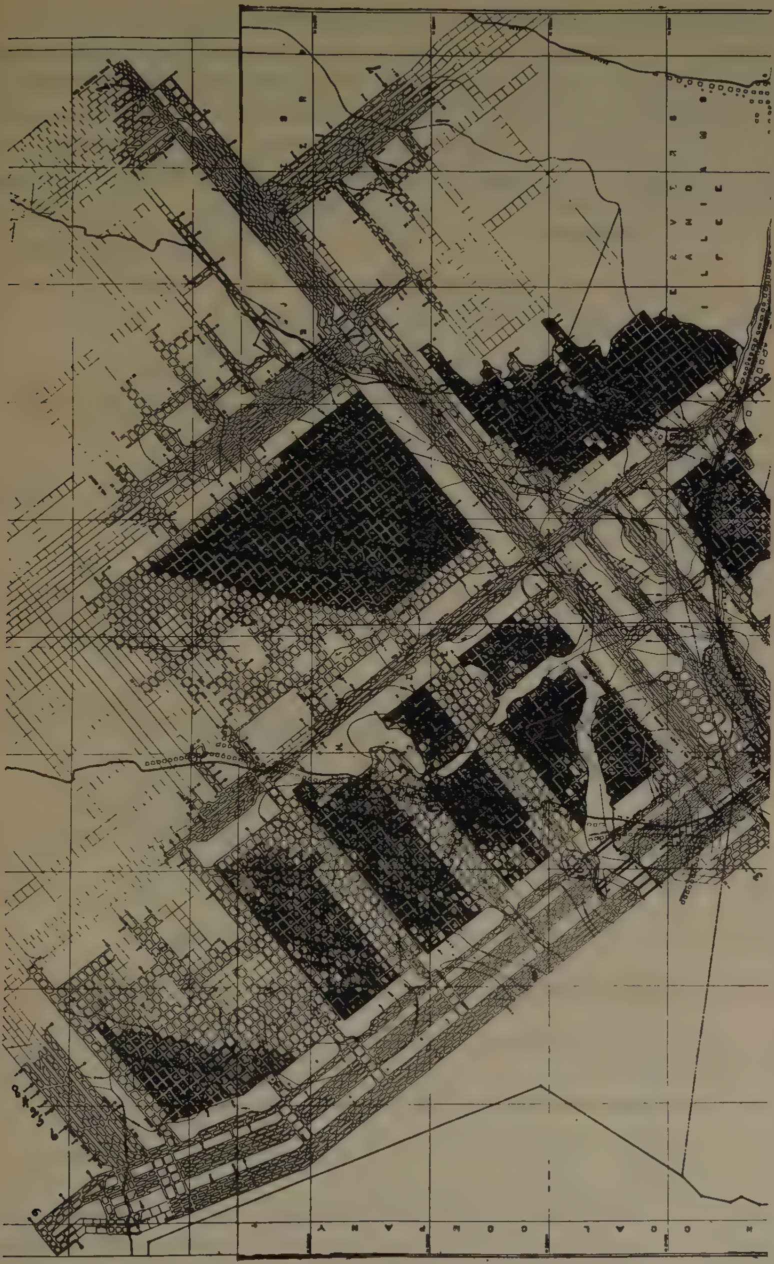

FIG. 4.—WORKINGS AT MINE A.
Station 1 at upper right-hand corner: 7, middle right edge; 3, lower edge, left of center; other stations, upper left.

Pressure and flow readings over a period of 49 days are shown graphically in Fig. 5. Readings on the holes in the roof and floor did not show any liberation of gas during this period. These holes did not contain any water. The quantity of water flowing from the hole in the coal bed declined throughout this period, and flow of gas also decreased. The pressure readings were taken after the valve to the hole had been closed about 20 min., when the pressure gage indicated that an apparent constant value had been reached. These are not true maximum pressures, however, because later experiments proved that these readings were approximately one-half of the true maximum pressures.

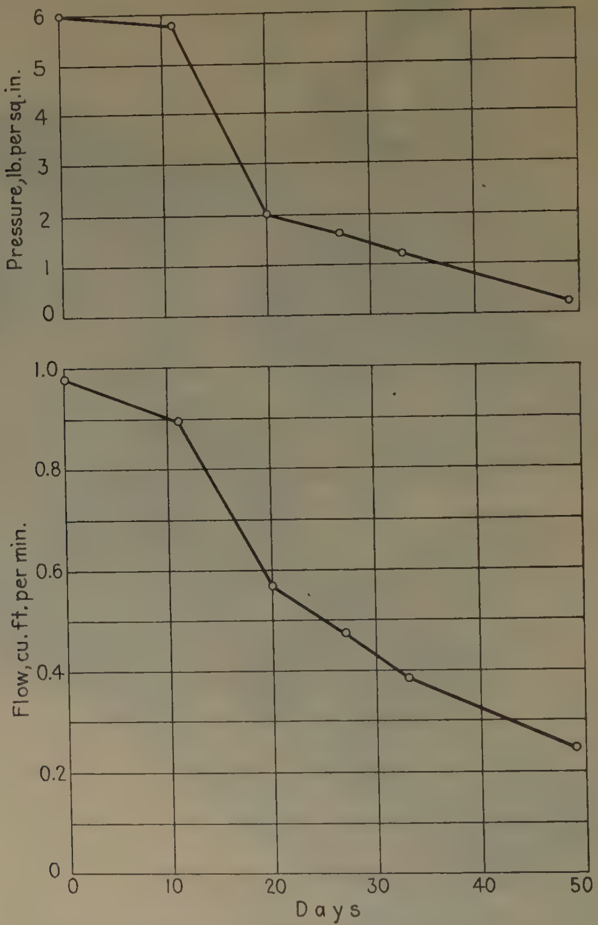

FIG. 5.—FLOW AND PRESSURE OF GAS AT STATION 1.

Station No. 2.—At experimental station No. 2 (Fig. 4) three horizontal holes, one above the other, were drilled into the coal bed and two vertical holes were drilled, one in the roof and one in the floor.

A hole, 1⅞ in. in diameter, was drilled horizontally 21 ft. deep into the center of the coal bed. At a depth of 12 ft. there was sufficient gas liberated to snuff out the flame of a safety lamp. When the hole was 16 ft. deep gas was noticed blowing the dry coal dust out of the hole; the volume of gas liberated increased until drilling stopped. The hole was cased to a depth of 6 ft. by inserting a ½-in. pipe and packing oakum around it. This hole remained dry throughout the period of observation. Pressure and flow readings were taken.

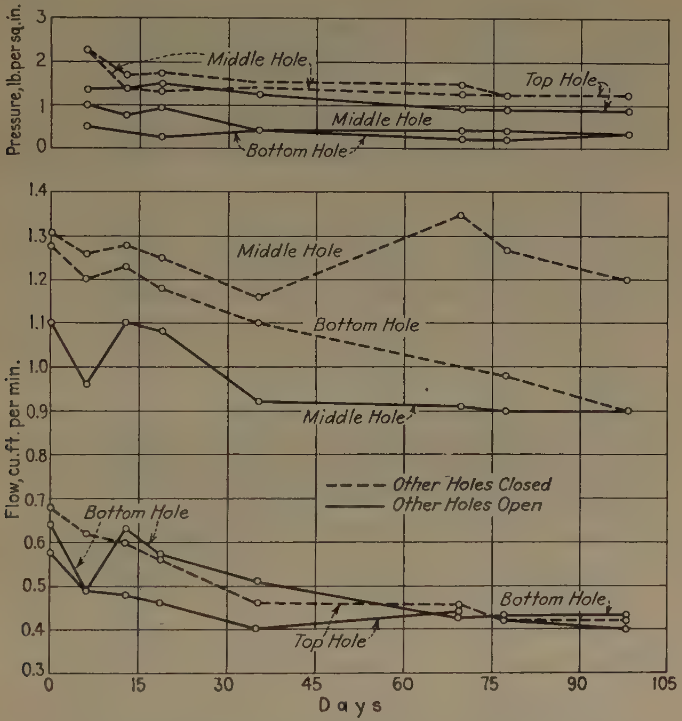

FIG. 6.—FLOW AND PRESSURE OF GAS AT STATION 2.

A hole 1⅞ in. in diameter was drilled horizontally 21 ft. into the coal and 13 in. from the bottom. The hole was cased to a depth of 6 ft. with ½-in. pipe. Pressure and flow readings were taken.

A hole 1⅞ in. in diameter was drilled horizontally 21 ft. into the coal and 9 in. from the top of the bed. The hole was cased with ½-in. pipe to a depth of 6 ft. Pressure and flow readings were recorded.

The pressure and volume of gas liberated from any one of the three holes were increased considerably when the other holes were closed, and decreased when the other holes were opened. The middle and bottom holes responded instantly to these changes. There was only a slight change on the pressure and volume of gas liberated from the top hole when the other holes were opened and closed; probably this was due to the fact that the gas had to pass through the shale parting that separated the top hole from the others.

A hole 1⅞ in. in diameter was drilled 6 ft. vertically into the roof. The hole was cased with ½-in. pipe to a depth of 5 ft. The initial reading showed a gas liberation of 0.035 cu. ft. per minute; no flow of gas was recorded thereafter. Throughout the mine this was the only flow of gas ever detected in holes that were drilled into the roof strata.

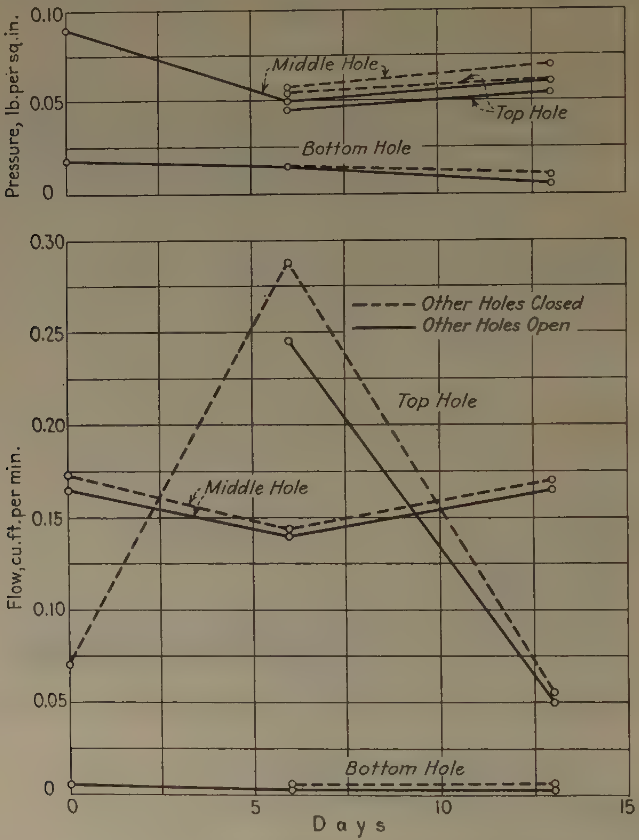

FIG. 7.—FLOW AND PRESSURE OF GAS AT STATION 3.

A hole 1⅞ in. in diameter was drilled 6 ft. vertically into the floor. The hole was cased with ½-in. pipe to a depth of 5 ft. No gas was detected.

All pressure and flow readings are shown graphically in Fig. 6.

Station No. 3.—At experimental station 3 (Fig. 4), three horizontal holes 1⅞ in. in diameter were drilled 21 ft. into the coal bed, one above the other, and holes were drilled into the roof and floor in exactly the same manner as at station 2.

The top hole was drilled approximately 8 in. below the roof and 1 in. above the top parting; the middle hole was drilled 1 in. below the middle parting, or 5 ft. above the floor; the bottom hole was drilled 1 ft. 11 in. above the floor. Each hole was cased to a depth of 9 ft. with ¾ in. iron pipe with oakum packing around it. The holes were free from water. Pressure and flow readings were taken, and the holes left open to drain. The gas pressure was low and the volume of gas liberated was small at this station; this was anticipated because the coal had been exposed for seven years and the gas had had ample time to drain from it.

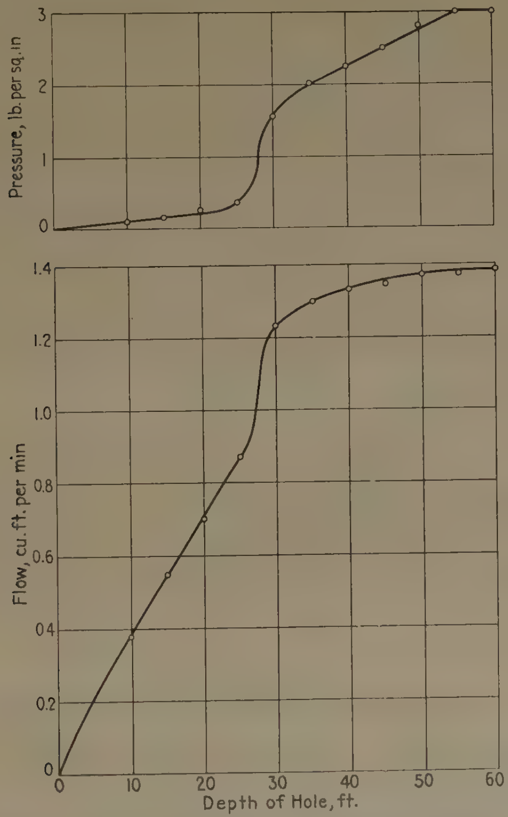

FIG. 8.—FLOW AND PRESSURE OF GAS AT STATION 4.

A hole 1⅞ in. in diameter was drilled 7 ft. vertically into the roof and was cased with ¾-in. iron pipe to a depth of 5 ft. There was no gas flow from this hole at any time during the period of observation.

A hole 1⅞ in. in diameter was drilled 8 ft. vertically into the floor and cased with ¾-in. pipe to a depth of 5 ft. Gas flow from this hole was never detected, even though the hole was closed to allow gas to accumulate. All pressure and flow readings taken at this station are shown graphically in Fig. 7.

Experiments to Measure the Flow and Pressure of the Gas

The experiments to determine the flow and pressures of the gas in the coal bed were divided into three groups as follows:

1. By means of advancing a horizontal hole to definite distances in the coal bed.

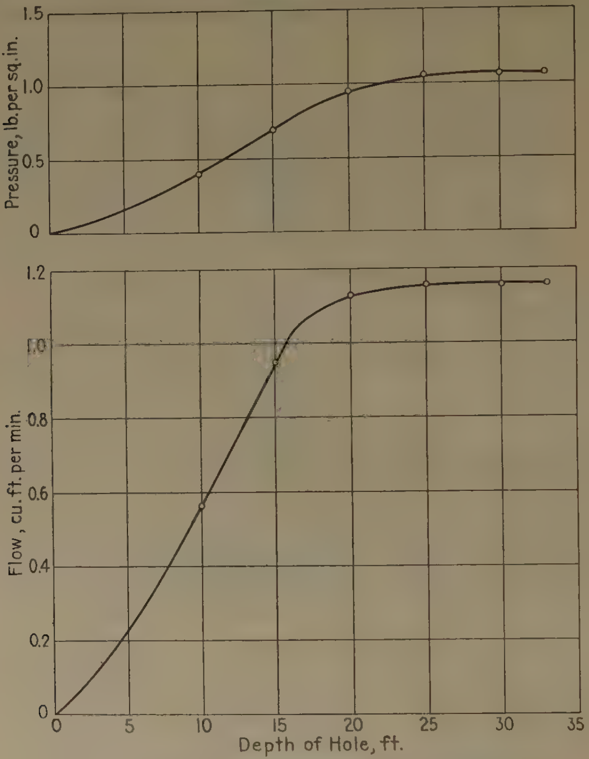

Fig. 9.—Flow and pressure of gas at station 5.

2. To obtain pressures of the gas at various depths in the solid coal by a series of horizontal holes cased and grouted.

3. To obtain some data on the flow of the gas from exposed coal faces.

Station No. 4.—A horizontal hole 3½ in. in diameter was drilled at location shown as No. 4 (Fig. 4). The hole was drilled 10 ft. into the coal bed, 6 in. below the bone parting. It was cased to a depth of 5 ft. with 2-in. iron pipe. Then pressure and flow readings were taken. The hole was then advanced by 5-ft. stages to a depth of 60 ft. 6 in. with flow and pressure readings taken at each stage. The results of these readings are shown graphically in Fig. 8.

Pressures were taken at each stage after the valve on the hole had been closed 20 min. These readings are apparent and therefore are not true maximum pressures.

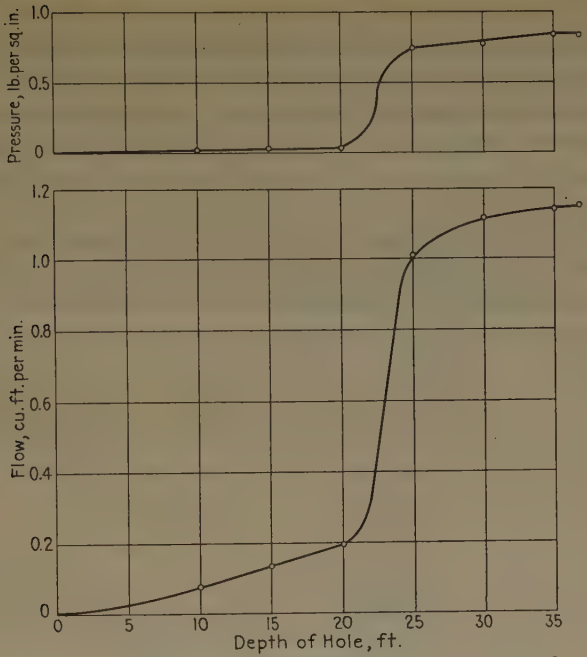

FIG. 10.—FLOW AND PRESSURE OF GAS AT STATION 6.

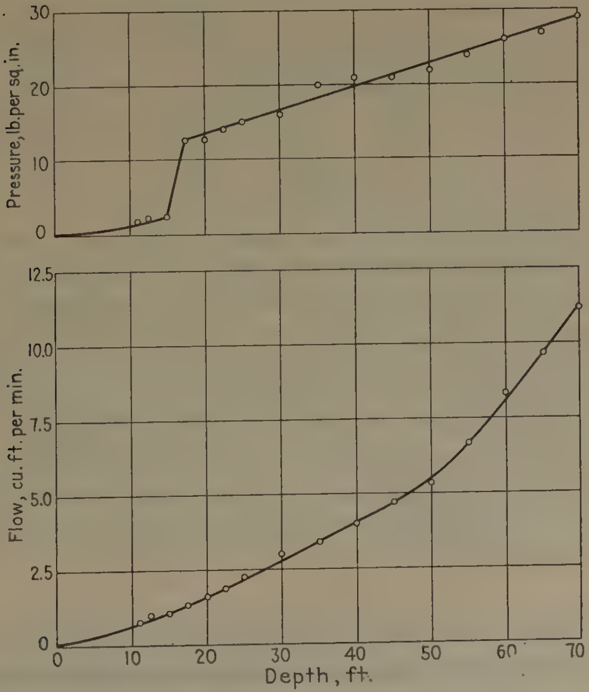

FIG. 11.—FLOW AND PRESSURE OF GAS AT STATION 7.

Station No. 5.—A hole 3½ in. in diameter was drilled horizontally 10 ft. into the coal, 5 in. below the bone parting at experimental station 5 (Fig. 4). The hole was cased to a depth of 6 ft. with 2-in. iron pipe. Pressure and flow readings were then taken. This hole was also advanced by 5-ft. stages to a depth of 33 ft. with flow and pressure readings taken at each stage. The results of these readings are shown graphically in Fig. 9.

Station No. 6.—At station 6 (Fig. 4) a horizontal hole 3½ in. in diameter was drilled 10 ft. into the coal bed and cased to a depth of 5 ft. with 2-in. iron pipe; then pressure and flow readings were taken. The hole was then advanced by 5-ft. stages to a depth of 36 ft. 9 in., where an impenetrable material was encountered. Flow and pressure readings were taken at each stage and results are shown graphically in Fig. 10.

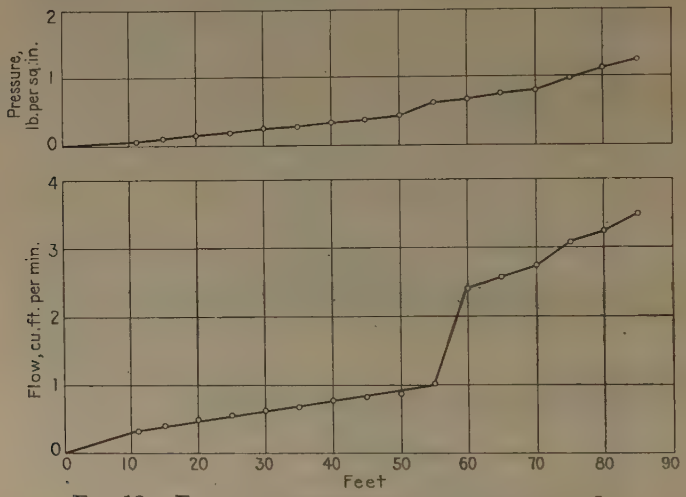

FIG. 12.—FLOW AND PRESSURE OF GAS AT STATION 8.

Station No. 7.—A 3½-in. hole was drilled 10 ft. horizontally into the coal bed at station 7 (Fig. 4), and cased in the same manner as the three preceding holes. It was also advanced in 5-ft. stages to a depth of 71 ft. 6 in. and pressure and flow readings were taken at each stage. The results of these readings are shown graphically in Fig. 11.

Station No. 8.—The hole at station 8 (Fig. 4) was drilled in the same manner as the four preceding holes. It was drilled into the coal bed to a depth of 85 ft. Pressure and flow readings taken at each 5-ft. stage are shown graphically in Fig. 12.

Gas Pressures at Various Depths in Solid Coal

Station No. 9.—This experimental station was located at the face of the north main headings shown as No. 9 on Fig. 4 and in detail in Fig. 13.

Ten holes each 4 in. in diameter were drilled horizontally into the center of the coal face at depths ranging from 15 to 102 ft. The location of these holes was confined to as small an area as possible to minimize any variation in conditions between them. It was necessary to drill the holes opposite crosscuts in order to have necessary room for drilling and casing.

After a hole was drilled a 2-in. iron pipe was pushed into it until the pipe was approximately 12 in. from the back of the hole. Grout was then pumped through the pipe until the space between the pipe and the walls of the hole was entirely filled. After the grouting process was completed the pipe was pushed to the back of the hole, tight against the solid coal, and the grout left in the pipe until it began to harden, then it was drilled out. After the grout had set for about 60 hr. the hole was drilled 1 ft. farther into fresh coal, the pipe was capped and the valve closed and locked.

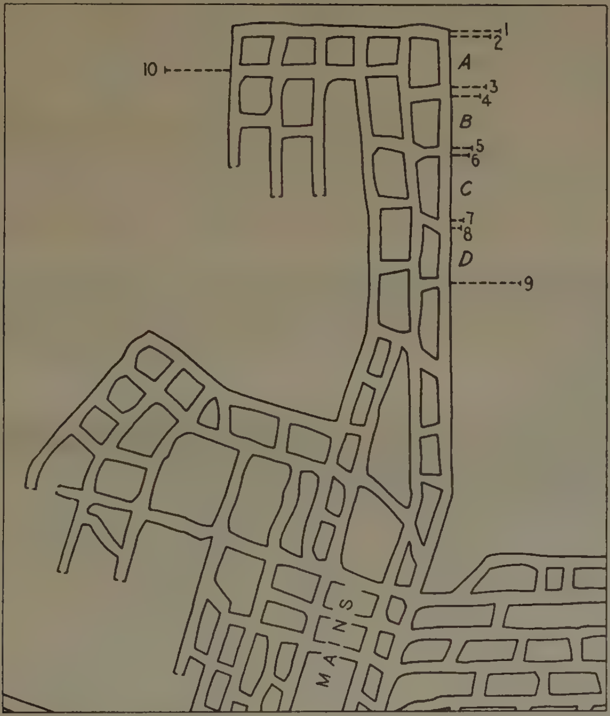

Fig. 13.—Location of holes at station 9.

Pressure readings were then taken on each hole over a period of 113 days, of which the results are shown graphically in Fig. 14. The large curve is plotted on depth versus maximum gas pressure in each hole while the small curves show gas pressure versus time in days for each hole;

thus the small curves show the history of the gas pressure over a period of days. Table 1 shows the average gas pressure in each hole. The fact that the gas pressures in holes 9 and 10, which were deeper than the others, are comparatively low indicates that location is a paramount factor in the gas pressures that exist in the solid coal at any one place. If the location of the first eight holes had been reversed, thereby placing

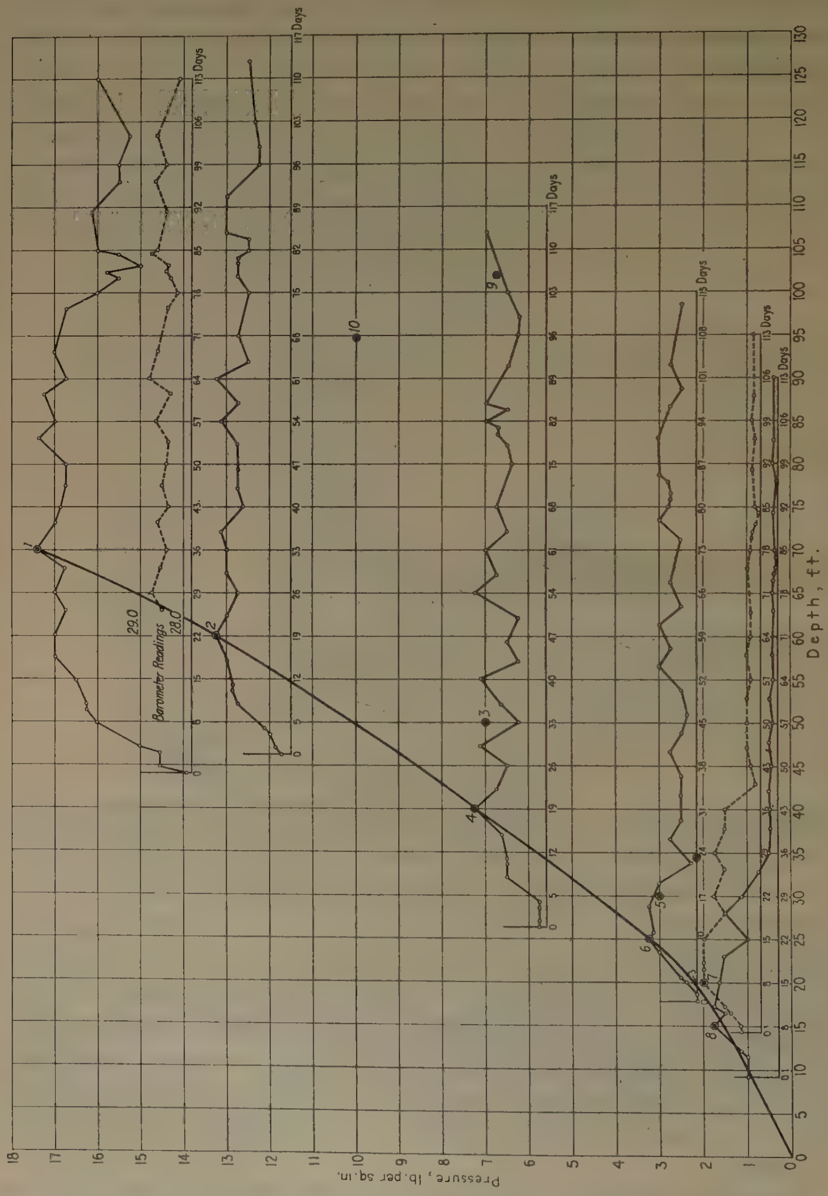

FIG. 14.—GAS PRESSURES AT STATION 9.

TABLE 1.—*Gas Pressure at Station* 9

Hole No.	Depth of Hole, Ft.	Distance of Hole from Coal Face, Ft.	Average Gas Pressure, Lb. per Sq. In.
1	70	3	17
2	60	15	13
3	50	89	6½
4	40	97½	6⅝
5	30	175½	2⅝
6	25	186½	2¾
7	20	272	1
8	15	283	½
9	102	363	6¼
10	95	60	9

the hole 15 ft. deep next to the coal face, the hole 20 ft. deep next, and so on, the depth versus maximum-pressure curve would probably have had a different slope. The shallow-hole pressures under this later arrangement would then be expected to be higher and the deeper hole pressures lower. The maximum gas pressure of 6¼ lb. per sq. in. in the hole that was 102 ft. deep would then probably lie approximately on the curve. These observations would lead one to believe that pressure intensity cannot be predicted accurately. In general it may be said that pressures increase as solid coal is penetrated and probably reaches a maximum some distance from the face. The observations show that gas is irregularly distributed throughout the coal bed, because of numerous

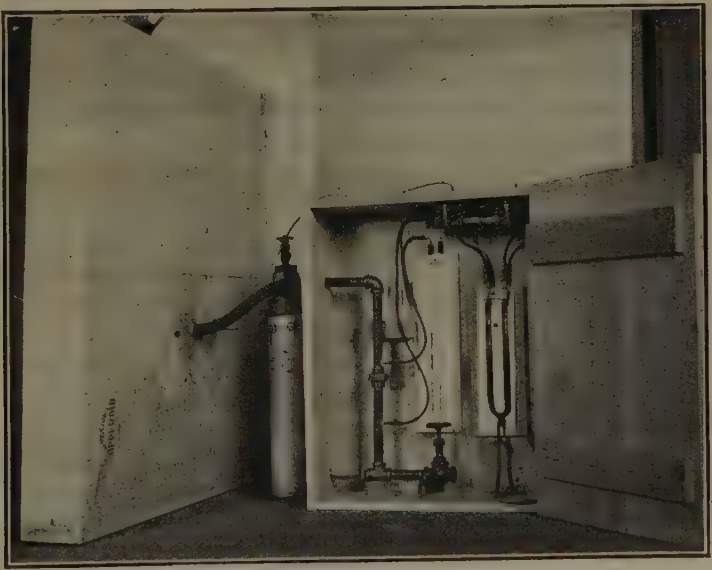

FIG. 15.—APPARATUS FOR MEASURING FLOW OF GAS.

conditions such as joint planes, slips and fractures, and changes in the physical character of the coal from place to place.

Flow of Gas from Exposed Coal Faces

Gas and flow readings from exposed faces of coal were measured at stations *A*, *B*, *C*, *D* (Fig. 13). These measurements were made by means of the apparatus illustrated in Fig. 15, which consisted of a galvanized iron shield 4 by 4 ft. by 4 in. deep, connected to an orifice meter and also to a differential pressure gage, by means of which the pressure inside the shield, when it was mounted in the coal face, could be observed.

In making gas-flow measurements it was necessary to equalize the pressures inside and outside the shield so that the gas flow lines would not be disturbed. This was done by aspirating oxygen gas into the gas-discharge line on the outby side of the orifice meter until the differential gage connected to the shield read zero, at which time orifice meter readings were taken. As these experiments are not yet completed, no interpretation of the results is now possible.

GAS ANALYSES

Gas samples were collected at stations 1, 2, 3, 4 and 7. The samples were taken directly from the cased holes by water displacement. The results of the analyses are shown in Table 2.

TABLE 2.—*Analyses of Gas Samples*

Sample No.	Where Obtained	Analysis,[a] Per Cent				
		CO_2	O_2	CH_4	N_2	C_2N_6
1	Station 1	0.0	3.1	85.9	11.0	0.0
2	Station 2	0.0	2.1	91.0	6.9	0.0
3	Station 3	0.0	0.6	97.7	1.7	0.0
4	Station 4	0.0	3.3	85.6	11.1	0.0
5	Station 7	0.0	1.6	93.2	5.2	0.0

[a] Samples analyzed by the United States Bureau of Mines.

Typical analyses of natural gases produced from sands show varying percentages of ethane. The samples listed in Table 2 show no ethane present in the gas emitted from the coal bed in mine A. Greenwald[2] has shown that ethane replaces or releases an equivalent amount of methane in its passage through coal and that this process continues until the coal contains all the ethane it can hold, when the ethane is liberated. If the gas in the coal in this mine were coming from a sand reservoir the large volume of methane that has already been liberated at mine A would probably have saturated the coal with ethane, so that the present gas samples would contain some ethane.

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[2] G. S. Rice: Introductory Notes on Origin of Instantaneous Outbursts of Gas in Certain Coal Mines of Europe and Western Canada. *Trans.* A.I.M.E. (1931) **94**, Coal Div., 83.

It is not necessary, however, to look for the source of the gas outside of the coal bed because it has been shown that coal beds can hold in solution enormous quantities of gas.[3] Of course, the breaking of the strata in a mine often allows gas to enter from overlying or underlying gassy coal beds.

CHARACTERISTICS OF SOME GASSY MINES IN WEST VIRGINIA

The observations made in this study show that the quantity of methane liberated varies considerably in the same coal bed even within short distances, and is due to the rate of mining or breaking down the coal, length of time the coal surface has been exposed, proximity of surrounding mine workings and virgin coal, physical character of the coal and other factors.

A study showing the relation of the amount of methane liberated to the metamorphism that the coal has undergone, to the rate of mining of the coal, and to the amount of cover over the coal bed, was made from data collected at six mines representing the northern high-volatile and southern low-volatile coal fields of West Virginia. These mines liberate a total of 23 million cubic feet of inflammable gas every 24 hr. when working normally. The thickness of the coal beds represented in this group of mines ranges from 6 to 8 feet.

The beds having the thickest cover and lying below surface drainage level liberate the greatest volume of gas in mining. Probably these coal beds have been slowly emitting gas throughout a long geologic period, so that at present the gas contained in them is only a part of the total original volume.

It is obvious from the tonnage curve (Fig. 16) that in general the larger the tonnage of coal mined the greater the volume of gas liberated from the mine. The last point on this curve, showing a comparatively low output of coal and a high gas liberation, is accounted for by the concentration of workings and the rapid advancement of these workings into solid coal at this particular mine. Breaking down of coal in mining is a very important factor governing the rate at which gas is liberated. The relationship between the volume of gas liberated and the tons of coal mined throughout the life of mine A up to December, 1932, is shown in Fig. 17. The maximum and minimum lines indicate the highest and lowest daily liberation of gas during each month. The general increase in the volume of gas liberated from the time mine A was opened indicates that in a gassy coal bed as the mine workings ramify the volume of gas liberated will increase and follow in general, up to a certain point, the curve showing the area of coal exhausted. It will continue to follow this curve under normal working conditions until the older ribs no longer contribute the large volumes of gas to the total volume liberated that they did formerly. This curve clearly shows that the amount of gas liberated increases or decreases with the amount of coal mined. It also shows, however, that at times the liberation of methane may change for no apparent reason.

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[3] J. I. Graham: Permeability of Coal to Air or Gas and the Solubilities of Different Gases in Coal. *Trans.* Inst. Min. Engrs. (1916–19) **52**, 338.

H. Briggs: Characteristics of Outbursts of Gas in Mines. *Trans.* Inst. Min. Engrs. (1920–21) **61**, 119.

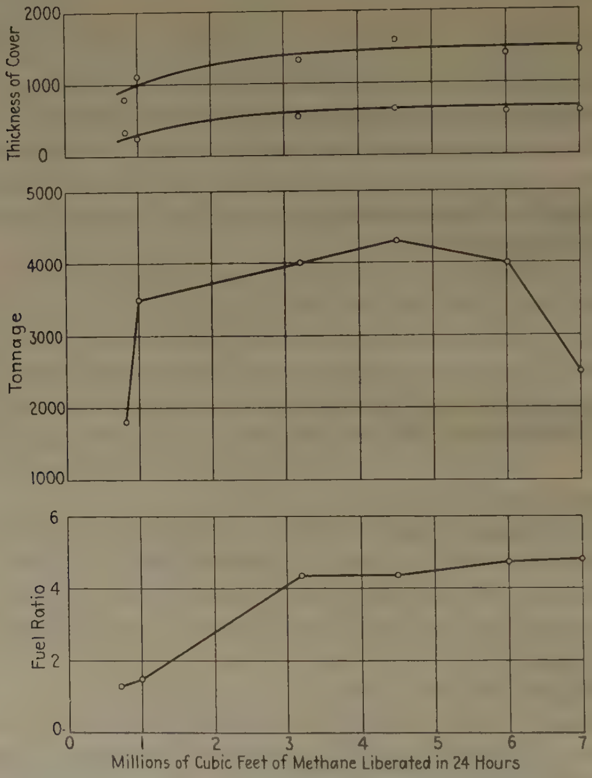

FIG. 16.—RELATION OF COVER, COAL MINED AND FUEL RATIO (F.C./V.M.) TO VOLUME OF METHANE LIBERATED IN SIX MINES IN WEST VIRGINIA.

Chamberlin[4] found that the rate of emanation of gas was increased by crushing the coal. In his experiments the coal was crushed in an airtight receptacle connected with a vacuum pump. His summary is as follows:

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[4] R. T. Chamberlin: Notes on Explosive Mine Gases and Dusts. U. S. Geol. Survey *Bull.* 383 (1909) 35.

1. Reduction of pressure removes the gases from coal slowly—a matter of weeks and months.

2. Finely powdered, fresh coal gives off more methane during six months in a vacuum than the same amount of lump coal subjected to the same conditions. But presumably during a much longer period of time approximately the same volume of gas should be expected to come off from either type of coal.

3. Reduction of pressure, acting for only a few hours or even a few days, is far less effective in extracting the entrapped gas than crushing the coal, either in a vacuum or under barometric pressure.

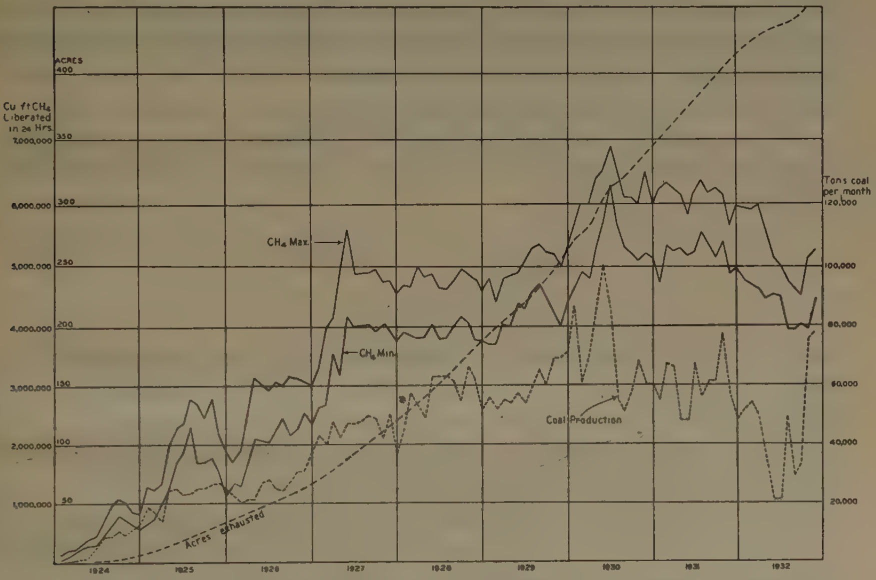

FIG. 17.—RELATIONSHIP BETWEEN TONS OF COAL MINED AND VOLUME OF INFLAMMABLE GAS LIBERATED IN MINE A.

4. Crushing coal to a certain degree of fineness yields approximately the same volume of methane whether the operation is performed in a vacuum or in an atmosphere of air or carbon dioxide.

The fuel ratio $\frac{\text{F.C.}}{\text{V.M.}}$ curve in Fig. 15 indicates that metamorphism has devolatilized the coals and thus possibly increased their gas content. If this happened a large quantity of gas would be anticipated in opening mines in porous, friable, low-volatile coals under thick and impervious cover, such as the Pocahontas coal bed No. 4 at mine A, where all these conditions are present.

SUMMARY

1. The study indicates that the large volume of gas in this coal bed probably has been generated within the coal bed; because the study has shown no evidence that the gas has migrated into the coal bed.

2. The gas in the coal bed probably occurs as a film over the the coal surfaces existing as joints and interstices in the coal, and also in the coal itself in an unstable condition so as to be readily released in mining.

3. In the mine where our observations were made, the roof strata or floor material, exclusive of coal beds, contains little or no gas.

4. To determine the maximum gas pressure in the Pocahontas No. 4 coal bed, it would probably be necessary to drill holes from the surface into the virgin coal at a place beyond the gas-drainage zone.

5. The liberation of gas in this coal bed increases with an increase of coal faces and ribs up to a time when the ribs that have been exposed for a long period no longer drain off much gas.

Partings in Coal Beds

By Albert W. Giles*, Fayetteville, Ark.

The splitting of the coal by partings is a familiar feature of many coal beds. The partings are normally argillaceous, more rarely sandy or calcareous. The material is usually fine-grained, less frequently coarse-grained to conglomeratic. Many partings are wedge-shaped, others are lenticular, and in some coal beds thin partings but one or two inches thick persist as thin sheets throughout areas of regional extent. The origin of the partings differs, and in some cases is obscure and uncertain.

ORIGIN OF THICK PARTINGS

Wedge-shaped Partings

Wedge-shaped partings are normally of indefinite linear extent but of short transverse section. They are developed adjacent to the borders of the original peat-forming swamp or around islands formed within the swamp, hence their presence is indicative of the approximate limit of the coal bed or of an interruption in the continuity of the coal bed. Following deposition of the parting gradual reestablishment of swamp conditions is followed by burial of the parting with accumulating peat.

Partings developed adjacent to the borders of swamps normally thicken toward the periphery of the swamp and represent either detrital materials carried in from adjacent land areas during floods or materials carried in by advancing waves and currents after a barrier confining the swamp had been breached by the waves. Partings formed in the manner first mentioned should become coarser and more arenaceous toward the source of supply of the detrital materials, and the overlying and underlying coal should gradually grade into carbonaceous shale as the margin of the original swamp is approached. Partings formed by advance of waves across a barrier should be largely or entirely arenaceous, thin wedges of short longitudinal section, possibly with fossils of marine or brackish affinity. The subjacent coal normally should "pinch out" against the barrier, and the overlying coal continue to the barrier, or even extend across it, depending on the geographic conditions established following the breaching of the original barrier.

Islands may be formed in swamps in a number of ways. They may represent local uplift of the swamp floor during earthquakes, or they may result from differential subsidence. The lowering of the water surface due to prolonged evaporation during periods of diminished precipitation may form islands of the higher parts of the swamp floor, which are exposed to the action of flood waters when rainfall increases. Large areas of flat country may be inundated by the formation of dams followed by peat formation. The breaking of a dam results in the lowering of the water level, the higher parts of the swamp becoming islands. Partings developed around islands would radiate outward for short distances from the islands. Their subsequent burial by peat following the resumption of swamp conditions would preserve them as wedge-shaped partings with flat bases.

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* Professor of Geology, University of Arkansas.

Lenticular Partings

Lenticular partings normally consist of clastic material and may originate in a number of ways. A stream draining into a swamp may deposit an alluvial fan on the surface of the peat. A parting originating in this manner would have a flat base, and a convex upper surface of low to imperceptible curvature. The material composing the accumulation would consist of a mixture of argillaceous and arenaceous sediment increasing in coarseness and arenaceous content toward the mouth of the feeding stream. Cessation of constructional activity by the stream, due to climatic or geographic changes, followed by gradual depression of the swamp area, would lead to the bridging of the fan by peat accumulation forming the coal above the parting.

Flooding due to excessive precipitation on surfaces draining into a swamp might readily result in the formation of a plexus of coalescent fans lying upon a peat base and thinning away from the shore to an irregular periphery. The recurrence of normal swamp conditions would lead to the burial of the compound fan by peat, the fan appearing in the subsequent coal bed as a parting with a flat base but undulatory upper surface.

Similar lenticular partings might develop on the inside of barriers following local breaking of the barriers by waves. The resulting partings would be chiefly arenaceous, possibly fossiliferous, of scarcely distinguishable lenticular shape, and of limited area. The healing of the breach by wave action would lead to the resumption of conditions favorable to peat formation and the deposition of peat over the surface of the sandy lens.

Since coal beds have for the most part accumulated in basins subject to continuous or intermittent subsidence, it seems reasonable to infer that in basins of significant extent differential subsidence must have operated at times, the results of which would be reflected in the physical features of the coal beds. In some cases the subsidence would not be of sufficient magnitude to interrupt the formation of peat, hence its result would be a coal bed possessing increased thickness where the subsidence occurred. In other cases the sagging would be sufficient to drown the flora resulting in a local interruption of peat accumulation which would continue until sedimentation had restored the floor of the sag to a level favorable for the reestablishment of the coal-forming plants. The building up of the floor of the sag might be due to inwash of fragments of peaty material which would form a thick bed of impure coal, or it might be due to filling with spores and macerated plant debris forming an impure cannel, or again the filling might be accomplished by inwash of sediment reestablishing swamp conditions. The sedimentary lens so produced would be convex on its lower side and relatively flat with peat cover on its upper surface. According to Kendall, "interruptions due to such an influence operating on a small scale are relatively rare." They are "commonly upon a regional scale and may mark the approximate limits of a seam."[1]

ORIGIN OF THIN PARTINGS

One or more thin partings, which may be coextensive with the coal bed, are characteristic of some coal beds. Such partings often maintain a remarkable uniformity in thickness, generally not exceeding 2 in., and frequently serve as a diagnostic feature in the recognition of the coal bed. They are normally fine grained and argillaceous.

The extensive distribution of many of these thin partings is a remarkable feature of the coal beds in which they occur. The parting in the Pittsburgh bed persists through several thousand square miles. The "blue band" in the No. 6 coal of Illinois has a thickness of about 1 in. and is known over several thousand square miles. The "flint-clay" parting characteristic of one of the Pottsville beds extends from central West Virginia into northern Tennessee. One or more coal beds of most coal fields usually carry similar thin and persistent partings.

The origin of the thin, persistent partings of coal beds has received some attention, and several explanations have been advanced to account for them.[2] There are objections to every explanation so far offered, and it is probable that no one general explanation is applicable in all cases.

David White has suggested that some of the partings may be of marine origin, particularly in coal beds formed in coastal swamps. E. S. Moore has suggested that in times of high water some streams laden with fine sediment have broken levees and flowed over the swamps, spreading the sediment over large areas. Since many of the coal swamps were essentially vast deltas lying at or but slightly above sea level, it seems conceivable that the streams draining into the swamps would develop a system of distributaries comparable to that of a large delta. The fracturing of the levees along distributaries would inundate adjacent swamp areas, the sediment of the muddy waters gradually settling to form the parting.

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[1] J. W. Evans and C. J. Stubblefield: Handbook of the Geology of Great Britain, 263. London, 1929. Thomas Murby & Co.

[2] D. White: Environmental Conditions of Deposition of Coal. *Trans.* A.I.M.E. (1925) **71,** 1–35.

The origin of thin, persistent partings in coal beds has also been regarded as due to erosion of low-lying lands adjacent to the swamps during times of excessive precipitation, the sediment being carried into the swamps by the flood waters. That the partings have resulted from chemical precipitation has been suggested, although this conception is not applicable except to those partings of calcareous composition, the origin of which would in general seem to be more like that of marl than as a direct chemical precipitate.

Reger has suggested that thin partings are formed by rapid evaporation of muddy water that invaded the peat-forming swamp, the suspended sediment rapidly settling to form the thin parting. Since the partings normally consist of materials sufficiently fine grained to be transported by air currents of moderate velocity, Thiessen has suggested that some of the partings may be wind-borne deposits of volcanic eruptions.

Ashley has suggested as a solution of the problem of the origin of thin, persistent partings that whirlwinds sweeping over practically desert areas lifted the matter very high in the air and allowed it to drift over and settle on the large swamp areas. Winds undoubtedly are effective transporting agents, carrying enormous quantities of dust in brief intervals of time. Their transporting activity is best illustrated in arid regions, but it is by no means confined to such regions. For several years Ehrenberg collected records of extensive deposits of atmospheric dust and in 1849 he published a map showing their distribution. His map indicates that extensive dust deposits occur in the greater part of the world.

Upper Banner Coal Bed Parting

For the purpose of testing the possible application of the several suggestions offered to explain the origin of thin, persistent partings in coal beds, a thin parting widely distributed in the Upper Banner coal bed of the southwest Virginia coal fields was selected for study.[3] This parting is not only one of the most persistent partings in the region but it is found in one of the most widely exploited coal beds of southwest Virginia; hence its characteristics are well known. The parting is fine grained, usually referred to as the "Upper Banner sandstone parting," and is present near the middle of the coal bed. The Upper Banner bed underlies large parts of Buchanan, Dickenson, Russell and Wise counties, having an area roughly estimated at 550 square miles (Fig. 1).

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[3] Samples were supplied to the writer through the courtesy of Mr. W. D. Tyler, Vice President of the Clinchfield Coal Corporation, Dante, Virginia.

In Buchanan County the Upper Banner is not well known and nearly everywhere the bed is thin. It appears to be absent from large areas of the county.[4] The sandstone parting is reported to be 1 in. thick and was found 2 ft., 9 in. above the base of the bed in one measurement in the western part of the county. In the few other measurements made in the county the parting is either absent or included with other divisions of the bed.

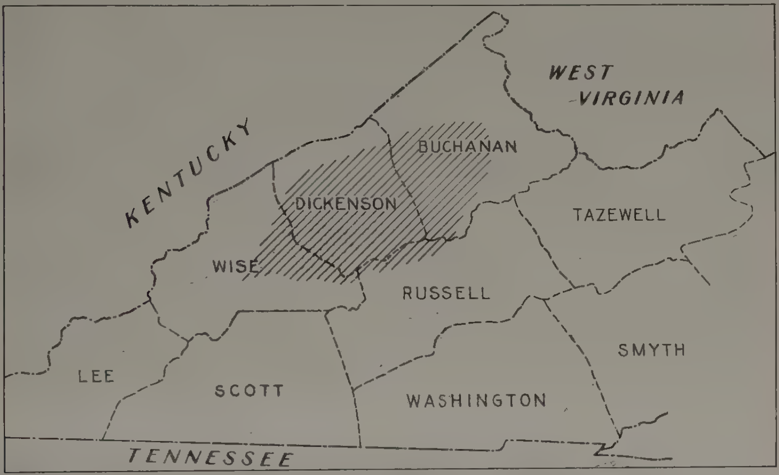

FIG. 1.—DISTRIBUTION OF UPPER BANNER COAL BED IN SOUTHWEST VIRGINIA.

The Upper Banner coal bed is well known in the north western part of Russell County.[5] The following statement describes the sandstone parting of the coal bed in this county: "A persistent sandstone parting less than 2 inches thick in the upper half of the bed is present in an area of several hundred square miles and is one of the most valuable identification characters known in any coal bed in the field." Every measurement of the Upper Banner bed given in the Russell County report includes the sandstone parting.

The Upper Banner bed is present in southern and central Dickenson County with "the persistent sandstone parting less than 2 in. thick in the upper half of the bed."[6] Every measurement of the scores of measurements of this bed in Dickenson County shows the presence of the sandstone parting except where the bed is split into two or more benches in the northwestern part of the county, where the sandstone may or may not be present in the upper bench.

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[4] H. Hinds: The Geology and Coal Resources of Buchanan County, Virginia. Va. Geol. Survey *Bull.* 18 (1918) 70.

[5] C. K. Wentworth: The Geology and Coal Resources of Russell County, Virginia. Va. Geol. Survey *Bull.* 22 (1922) 45.

[6] A. W. Giles: The Geology and Coal Resources of Dickenson County, Virginia. Va. Geol. Survey *Bull.* 21 (1921) 73–74.

The Upper Banner coal bed has been actively exploited in eastern Wise County, where "throughout much of its extent the Upper Banner is parted usually near the middle by a characteristic bed of sandstone 1 or 2 in. thick. At some localities it is divided by two or more partings, usually of the same type of hard, carbonaceous, sandstone."[7] Locally shale, "rash" or even bone may occur instead of the sandstone parting. These may contain grains of quartz and when traced laterally grade into the typical sandstone parting. The numerous measurements of this bed in Wise County indicate that the sandstone parting is as prevalent and characteristic as farther east in Dickenson and Russell counties.

Examination of representative samples of the Upper Banner parting with a hand lens reveals a fine-grained, compact, dark gray to black carbonaceous rock with the typical gritty feel of sandstone. The color is due to carbonaceous matter present. Thin seams of coal, ranging in thickness from that of a knife-edge to the diameter of a coarse thread, locally penetrate the sandstone both diagonally and parallel to the bedding.

Examination with the binocular of pieces selected as typical of the parting, magnification about 32x, reveals a confused mixture of coaly matter and mineral grains consisting chiefly of angular quartz grains, flakes and shreds of mica, and occasional small grains of pyrite or marcasite and rarely grains of feldspar in an aphanitic matrix. The quartz grains are angular, and consist of glassy and milky quartz rarely exhibiting etched or corroded surfaces. The feldspar grains are angular and dark gray and appear fresh under the binocular. The mica shreds and flakes show considerable alteration and the effects of weathering.

The coaly material consists of blebs of coal scattered through the matrix, and of thin, short seams and lenses of coal ramifying in various directions through the matrix. The blebs and seams of coal may contain mineral fragments. Much of the material is so soft that it can readily be cut with the point of a knife, and when scraped with the edge of the knife blade the surface of the sample appears gray and feels soft and smooth.

A number of thin sections representative of the parting were submitted to Dr. J. T. Stark for microscopic examination. His study indicates that the greater part of all the sections consists of fragmental, fibrous material, which appears to be organic. Many of the fragments show curved and vermicular shapes (Figs. 2 and 3), have weak pleochroism and weak birefringence. Lamination is suggested in the rough paral-

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[7] J. B. Eby: The Geology and Mineral Resources of Wise County, etc. Va. Geol. Survey *Bull.* 24 (1925) 149, 354, 370.

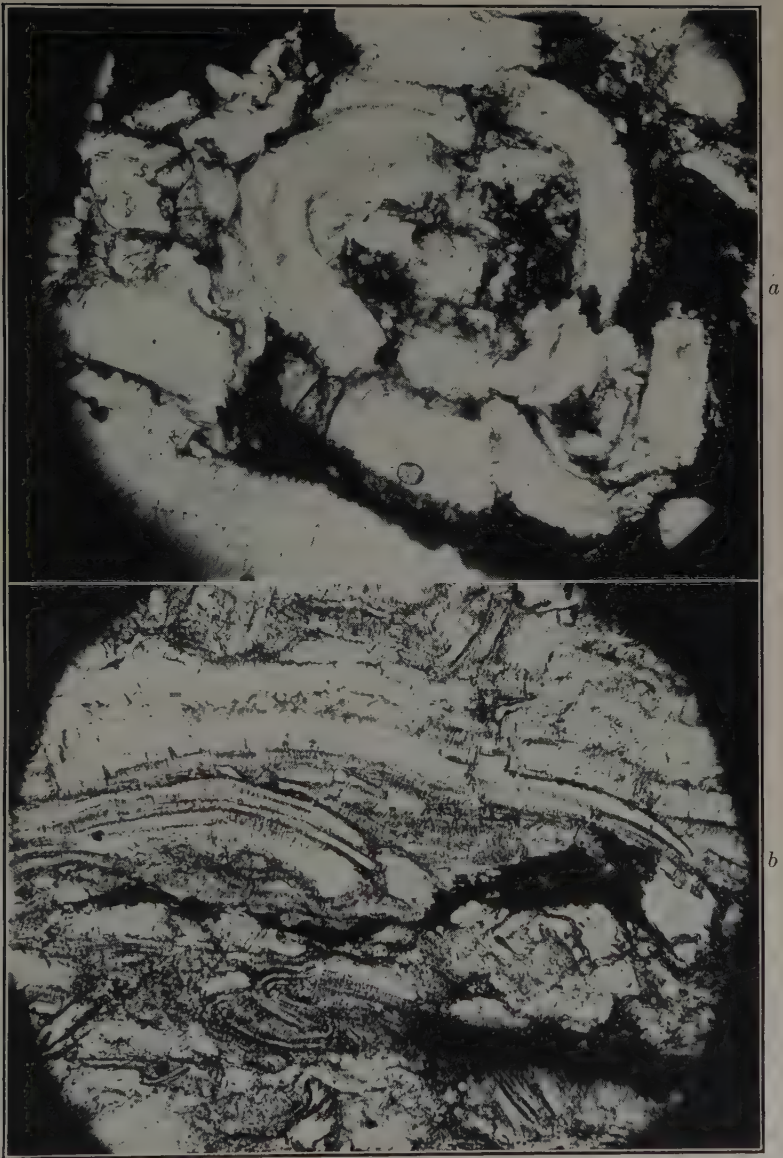

FIG 2.—PHOTOMICROGRAPHS OF THIN SECTIONS REPRESENTATIVE OF UPPER BANNER COAL-BED PARTING OF SOUTHWEST VIRGINIA. × 110.

a. Sample from mine No. 6, northwestern Russell County.

b. Sample from mine No. 1, northeastern Wise County.

Plane polarized light showing curved and vermicular shapes characteristic of the matrix.

Samples from Clinchfield Coal Corporation.

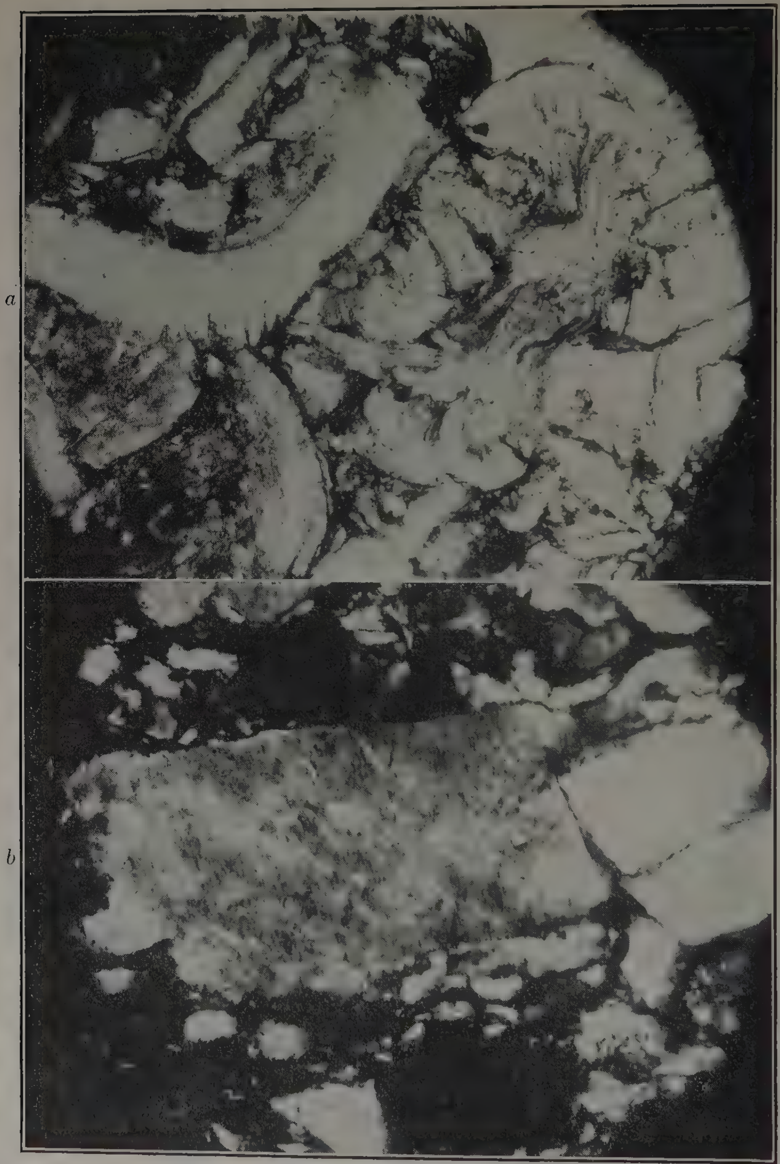

FIG. 3.—PHOTOMICROGRAPHS OF THIN SECTIONS REPRESENTATIVE OF UPPER BANNER COAL-BED PARTING OF SOUTHWEST VIRGINIA. × 110.

a. Sample from mine No. 7, north-central Dickenson County.

b. Sample from mine No. 3, northwestern Russell County.

a. Plane polarized light, showing curved and vermicular shapes characteristic of the matrix.

b. Crossed nicols, showing wavy extinction of large fragment of matrix material.

Samples from Clinchfield Coal Corporation.

lelism of the longer diameters of the fragments which average usually less than 1 mm. This lamination is more pronounced in the finer grained specimens.

The fragmental, fibrous material is more or less closely spaced in a fine detrital matrix, much of which is too fine for determination. It resembles dust or siltlike material common in fine sediments. The detrital material varies in amount from section to section. In some sections the larger fragments are very closely spaced while in others there is almost 40 per cent of finer detritus. In general the detrital matrix material varies between 10 and 35 per cent.

Quartz is the most conspicuous and abundant mineral in the fine detrital matrix. The grains range in size from very small particles to 0.5 mm. in length. They are sharply angular and show no indication of aqueous rounding. Notwithstanding their abundance in comparison with the other minerals present, the quartz grains make up but a small part of the sections, even where thickest less than 3 or 4 per cent of the section.

Feldspar grains were recognized with certainty in only one or two sections. The fragmental material presents no clear evidence of twinning and there is a definite absence of sericitization, kaolinization, and saussuritization, frequent in feldspar alteration. The fine detrital matrix may contain decomposed feldspathic material.

In several of the sections altered mica is present in the form of shreds and small flakes. The smaller flakes might be interpreted as sericite but much of the shredded mica appears to have been altered from biotite. In one section a larger flake was determined as biotite, showing the characteristic pleochroism and "bird's eye maple" appearance. In another section a fairly fresh flake of muscovite was recognized. In the hand specimens several whitish flakes suggest muscovite but are more likely bleached biotite. It is difficult because of the highly altered condition to estimate the percentage of mica but even where it is most abundant it is less than one per cent of any section.

In all of the sections there is a slightly opaque to dense, dark brown to black material occurring in large masses and veinlike stringers between the other constituents. This is a mixture of limonite and manganese oxide (probably wad). Examination of the sections for the presence of chlorite reveals little that is suggestive. It is certainly in negligible amount, although there is undoubtedly some chlorite in connection with the altered biotite shreds. One or two small grains of zircon were determined. Many small areas with alteration haloes are suggestive and may represent other zircon centers.[8]

Several thin sections of the Upper Banner parting were also submitted to Dr. Thiessen for examination. He reports that the major part of the rock consists of an inorganic matrix which behaves optically like a glass. In this matrix there are suspended a large number of small grains of mineral matter in crystal form. There are also embedded in the matrix: (1) organic matter in colloidal form in various concentrations so as to form irregular streaks of lighter and darker color, and (2) finely divided mineral matter or mineral matter in colloidal form dispersed through the matrix, also in various concentrations so as to form more or less regular streaks and figures, here and there heavy patches, and in other places very sparsely represented so as almost to form voids. The mineral matter in crystal form suspended in the matrix also holds colloidal mineral matter in suspension.[9]

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[8] Personal communication from Dr. J. T. Stark, dated Dec. 4, 1930.

Conclusion

The thin parting in the Upper Banner coal bed of southwest Virginia is a persistent layer characteristic of the coal bed, and very uniform in thickness and character throughout the extent of the coal bed. The identifiable primary minerals present—quartz, feldspar, biotite, muscovite and zircon—indicate derivation from a region underlain by acid igneous or metamorphic rocks, such as granite, quartz diorite, or gneiss. The chlorite, limonite, manganese oxide, etc., are secondary in origin. The source of the minerals and the fine materials comprising the matrix of the parting was in all probability the Piedmont Plateau, a region of igneous and metamorphic rocks. The angularity of the mineral grains denotes short distance transportation, and the freshness of many of the grains indicates mechanical weathering of the parent rock mass. The parting appears to be detrital in origin, deposition occurring during a brief and shallow inundation of the Upper Banner swamp, in which peat accumulation had continued for some time. The presence of thin seams and lenses of coal in the parting and the local transition of the parting into "rash" or "bone" indicate shallow submergence scarcely interrupting normal swamp conditions. Such shallow submergence was presumably of brief duration. A single period of flooding following excessive precipitation, or successive closely spaced floodings during a cycle of abnormally wet years, would afford ample time for the deposition of the parting. The muddy flood waters advancing from several places along the margin into the swamp, already more than normally inundated, following excessive precipitation, knocked down much of the partially submerged undergrowth in its advance and spread relatively unimpeded throughout the swamp, depositing the minerals and the fine matrix composing the thin parting of the coal bed. The wind may also have deposited in the shallow water covering the peat a small fraction of the fine material composing the matrix. No evidence of volcanic origin for any of the material is indicated.

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[9] Personal communication from Dr. R. Thiessen, dated April 7, 1932.

Further Notes on Bumps in No. 2 Mine, Springhill, Nova Scotia

By T. L. McCall,* Sydney, Nova Scotia

(New York Meeting, February, 1934)

The late Walter Herd[1] in 1929 gave a full description of past and present conditions in No. 2 mine, Springhill, Nova Scotia, developed theories regarding the cause of these bumps and made certain suggestions regarding possible methods of future development.

Recapitulating briefly from Mr. Herd's paper, the seam is bituminous coal, averages 9 ft. in thickness and has a well defined parting containing fusain about 14 in. from the roof. This roof coal is somewhat stronger than the rest of the seam and has a tendency to cling to the roof. The pitch of the seam where it is now being worked is about 20°.

The first manifestations of bumps began when the room and pillar workings had reached a depth of cover of 1900 ft. and under continuously increasing depth of cover these bumps increased to such an extent, both in number and severity, that in the year 1924 it became apparent that some change in the system of extraction was necessary. After closely studying all the factors involved and benefiting largely by valuable outside advice obtained by the Provincial Department of Mines of Nova Scotia,[2] the system of working was changed from room and pillar to longwall retreating.

Mr. Herd's paper described in detail three remarkable bumps in the year 1928 in April, July and October that affected the 5700-ft. and 5900-ft. East levels, the last bump being so severe both in intensity and in extent of damage done that this district was abandoned.

The longwall workings to the west of the slope had been stopped in the year 1926 owing to an inbreak of water, coming from the flooded waste of No. 3 seam lying some 400 ft. vertically overhead. This inflow was not excessive, but a large quantity of water being impounded in the old workings it was considered advisable to take the prudent course and not incur the risk of possibly increasing the flow by further working. This district has remained idle ever since.

Manuscript received at the office of the Institute Dec. 12, 1933.

* Chief Mining Engineer, Dominion Steel & Coal Corporation.

[1] W. Herd: Bumps in No. 2 Mine, Springhill, Nova Scotia. *Trans.* A.I.M.E. (1930) **88,** Coal Div., 151.

[2] G. S. Rice: Occurrence of Bumps in the Springhill No. 2 Mine. Report of Department of Public Works and Mines, Nova Scotia (1924).

The outcome of these restrictions has been to confine the workings of No. 2 mine to a district east of the main slope, this district lying between the extreme eastern limit of the mine and the area affected by bumps.

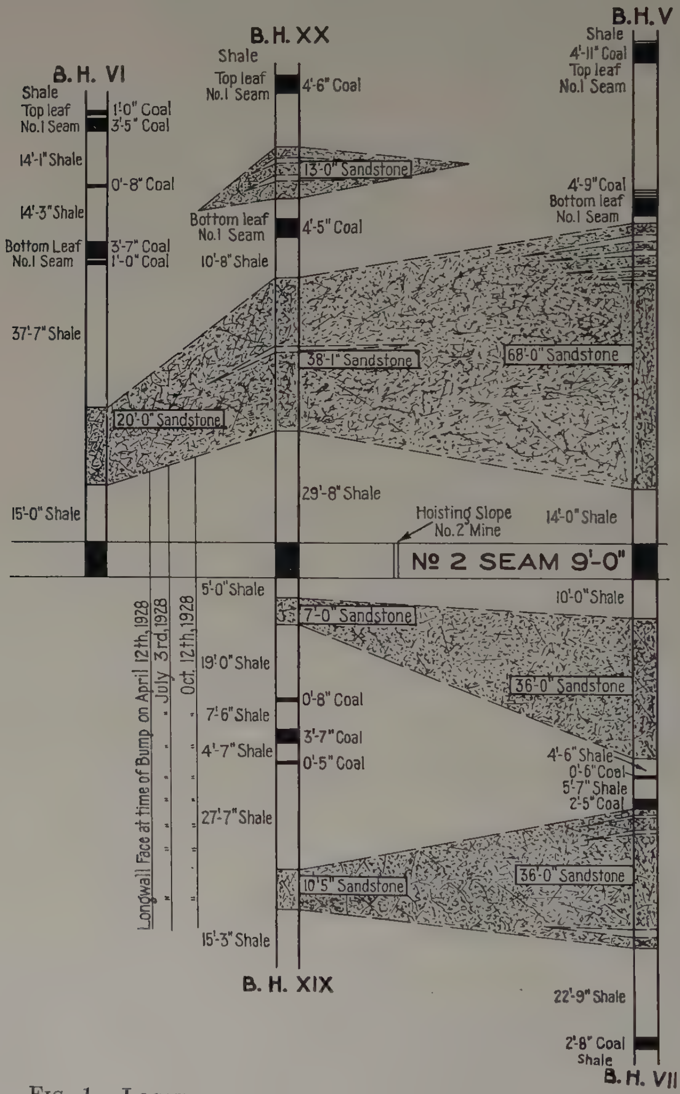


FIG. 1.—LOCATION OF BOREHOLES IN ROOF AND PAVEMENT.

BOREHOLES TO ASCERTAIN CHARACTER OF ROCKS

In recent years more boring has been done to ascertain the character of the rocks immediately above and below No. 2 seam in the zone subject to bumps. Boreholes XIX and XX (Fig. 1) were bored in the roof and pavement respectively at a point 650 ft. ahead of the place at which the east side longwall faces had been stopped. The relative positions of the longwall faces at the time of the three bumps referred to are shown on this section, from which it would appear by interpolation that the thicknesses of the sandstone band overlying the faces at the time of the bumps were 26, 29 and 32 ft., respectively.

There is no record of the nature of the pavement strata at the location of borehole VI, but to judge from the findings of B.H. XIX there can only have been a relatively thin band of sandstone present. However, the behavior of the pavement in this district, little if any motion having been apparent, indicated that it could be classified as a strong pavement.

The loss of output occasioned by abandoning the 5700-ft. and 5900-ft. east levels was replaced by establishing longwall faces at the extreme eastern limits of the 6300 and 6500-ft. levels, a district in which bumps are unknown. These new walls produced steadily, using the same methods of roof support as formerly, and in the year 1931 entered into the zone where the overlying sandstone bed, as proved by boring, began to increase in thickness.

CONTINUANCE OF BUMPS

When the 5700 and 5900-ft. longwall faces had reached this point, the ground in advance of the faces could be best described as being uneasy; small bumps occurring repeatedly in the roadways leading to the faces, dislodging coal from the high side ribs and occasionally knocking out timber.

When the longwall faces of the 6300 and 6500-ft. levels entered this area—all the coal being extracted up to the waste formed by the 5700 and 5900-ft. level walls—the ground gave fewer indications of uneasiness, minor bumps occurring less frequently, and hopes were entertained that the wider span of waste now prevailing would facilitate the fracturing of the sandstone bed, the "hanging up" of this bed over the waste, should such be the case, and its subsequent collapse, being regarded as a possible contributory cause of the bumps.

However, on Nov. 7, 1931, a severe high-side bump affected the 6500-ft. east level near the longwall face. At this point the continued loosening of the coal from the high side rib had widened the level to such an extent that wooden support packs had been built along the high side of the level.

The effects of the bump were first discernable at a point 130 ft. outside of the wall and extended out for a further distance of 90 ft. Coal from the high side rib was found to have been thrown down and the high side packs to have been demolished and thrown to the low side of the road. Eight empty boxes had been standing on the high side track and these together with the track had been violently hurled to the low side of the roadway, the track being torn up for a distance of 110 ft. The demolition of the high side packs and the displacement of the booms they supported permitted some loose roof stone to fall.

For two days prior to this bump the roof at this part of the level had been "working" slightly and small flakes of stone had been falling; this action ceased after the bump. On the longwalls the roof over the waste of the 6500-ft. level had been holding up between the midwalls for a distance of about 70 ft. back from the face into the waste. On the day previous to the bump heavy waste falls took place, which, as far as could be seen, completely closed the wastes between the midwalls. After the roadways had been cleaned up and put in order the walls resumed production and continued to advance, conditions being perfectly normal.

On Sunday, Nov. 22, 1931, at 1:05 p.m. an extremely heavy district bump occurred, which was felt on the surface and affected the main slopes, the 6500, 6900 and 7100-ft. levels, but none of the damage came within 1000 ft. of the longwall faces. On the same day another bump occurred at 10:10 p.m., which completely wrecked the 6300 and 6500-ft. levels, the damage in this instance being confined to the roadways within a distance of 1000 ft. from the longwall faces. The faces themselves were undamaged.

The thickness of the overlying sandstone band, to judge by the findings of neighboring boreholes, was 25 ft., a thickness that compares with the 26 ft. found on the 5700 and 5900-ft. levels at the time of the disastrous bump in April, 1928.

Bump at 1:05 *p.m., November* 22, 1931

The resultant effects of this bump covered an extensive territory, as may be seen by reference to Fig. 2, all the damage occasioned being to the west of No. 1 incline.

On the pipe slope coal was thrown off the east side rib from the 6500-ft. counter level to the 6800-ft. lodgment and several east side props were knocked out. Coal was thrown down from the high side rib in the roadway between the main and pipe slopes at the 6800-ft. lodgment and at this point 10 sets of high side props were knocked out.

On the main slope the east side ribs and props were thrown down between the 6900 and 7100-ft. levels and between the 6500-ft. level and the waiting head above it.

In the 6500-ft. waiting head both high and low side ribs were thrown down and some props were knocked out. On the 6500-ft. turnout some coal was thrown from the high side rib and some high side props were knocked out. The same thing happened in the crosscut below the 7100-ft. level, between the fan slope and the hoisting slope.

On the fan slope the east side rib was thrown down and a large number of east side props knocked out between the 6500-ft. low level and the 6900-ft. level. Lack of support permitted a considerable quantity of stone to fall. Between the 6900 and 7100-ft. levels coal was thrown from the east side rib. The 6500-ft. overcast was damaged. The walls were cracked and the roof was broken through.

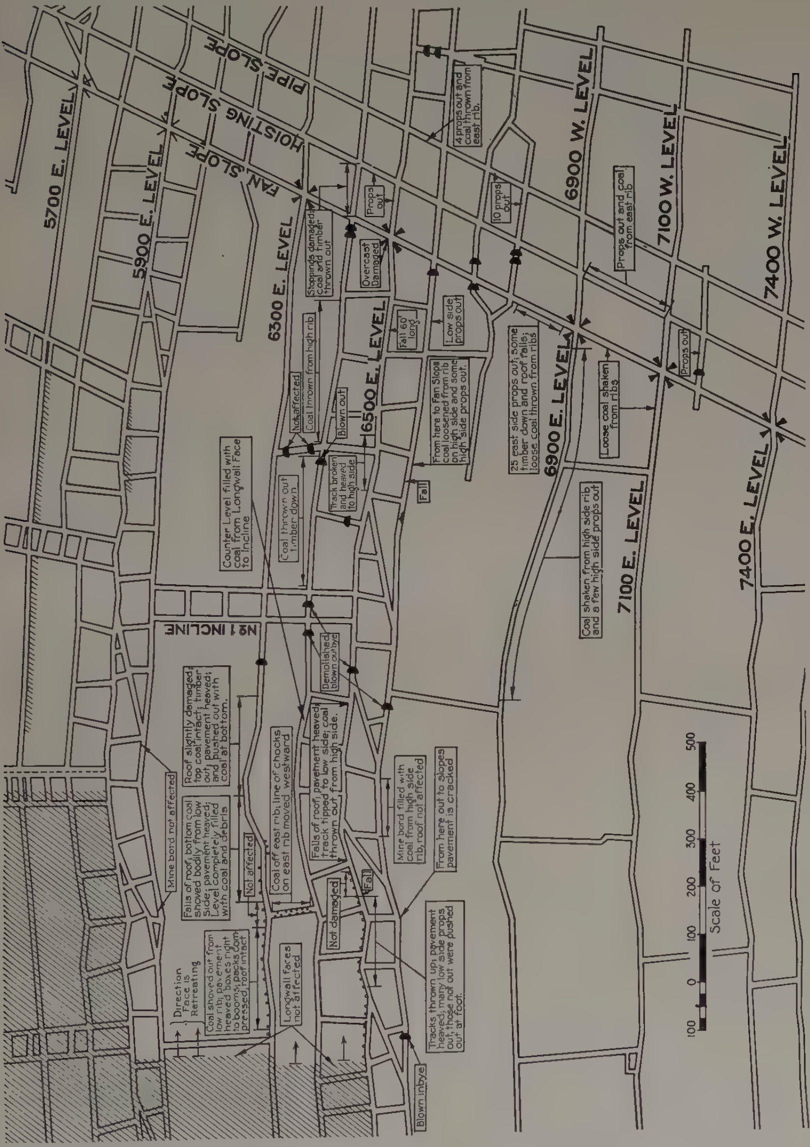

5700 E. LEVEL
5900 E. LEVEL
PIPE SLOPE
HOISTING SLOPE
FAN SLOPE
6300 E. LEVEL
6500 E. LEVEL
6900 W. LEVEL
6900 E. LEVEL
7100 W. LEVEL
7100 E. LEVEL
7400 W. LEVEL
7400 E. LEVEL
No 1 INCLINE
4 props out and coal thrown from east rib
Props out
10 props out
Overcast Damaged
Stoppings damaged; coal and timber thrown out
Props out and coal from east rib
Fall 60' long
Low side props out
25 east side props out; some timber down and roof falls; loose coal thrown from ribs
Loose coal shaken from ribs
Props out
Not affected
Coal thrown from high rib
Blown out
From here to Fan Slope coal loosened from rib on high side and some high side props out.
Track broken and heaved to high side
Fall
Coal thrown out timber down
Counter Level filled with coal from Longwall Face to Incline
Coal shaken from high side rib and a few high side props out
Demolished blown outbye
Roof slightly damaged; top coal intact; timber out; pavement heaved; and pushed out with coal at bottom.
Falls of roof, bottom coal shoved bodily from low side; pavement heaved; Level completely filled with coal and debris
Mine bord not affected
Not affected
Coal off east rib; line of chocks on east rib moved westward
Falls of roof, pavement heaved; track tipped to low side; coal thrown out from high side.
Mine bord filled with coal from high side rib, roof not affected
From here out to slopes pavement is cracked
Not damaged
Fall
Direction Face is Retreating
Coal shoved out from low rib; pavement heaved boxes right to booms; packs compressed, roof intact
Longwall faces not affected
Tracks thrown up; pavement heaved; many low side props out, those not out were pushed out at foot.
Blown inbye
Scale of Feet
100
0
100
200
300
400
500

FIG. 2.—PLAN SHOWING EFFECTS OF BUMPS OF NOVEMBER 22, 1931.

In the 6500-ft. counter level coal was thrown down from the high side rib and some high side props knocked out from a point 150 ft. inside the fan slope to No. 1 incline.

In the 6500-ft. level, beginning at a point 100 ft. inside the fan slope and continuing for a distance of 250 ft., coal was thrown down from the high side and the timbers were knocked out. A fall of stone 60 ft. long occurred at this point and closed the level. Forty feet inside this damage from the high side, the track was broken and thrown to high side for a length of 100 ft.

For 420 ft. east of the fan slope some coal was thrown down from the high side rib of the 6500-ft. low level and some high side props were knocked out. For 60 ft. east of the fan slope coal and some props were thrown from the low side rib of this mine bord. In the 6900-ft. level some coal and a few props were thrown from the high side over a length of 800 ft. east of the fan slope. In the 7100-ft. level loose coal was shaken from the high side rib for 140 ft. east of the fan slope.

Immediately after this bump the mine was examined and, apart from the damage noted above, conditions were found to be normal and the longwall faces absolutely undisturbed.

It is of interest to observe from the plan that on the roadways paralleling the strike only in one instance—on the 6500-ft. level—was the force exerted from the low side of a roadway.

Bump at 10:10 *p.m., November* 22, 1931

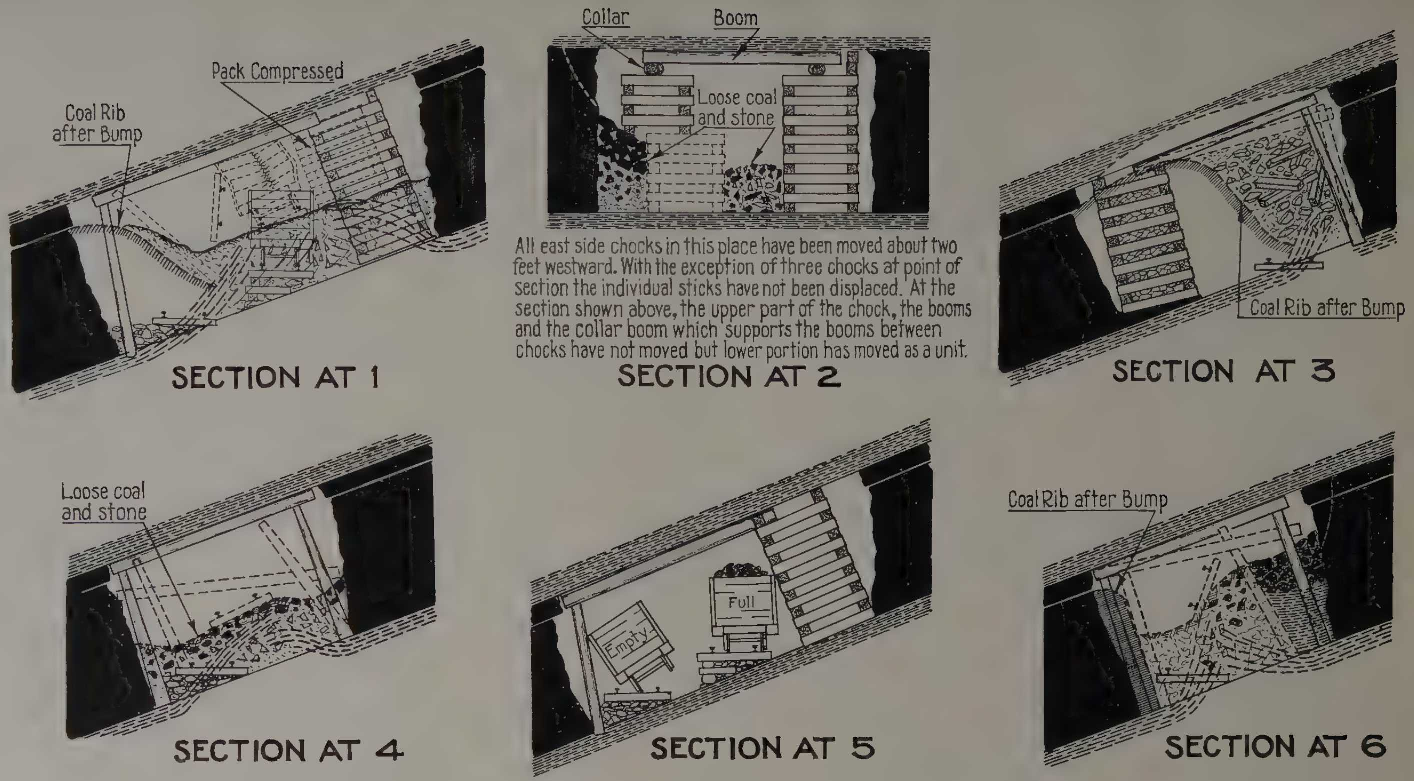
The havoc wrought by this bump has also been indicated on the same plan (Fig. 2) but in this instance all the damage occurred to the east of No. 1 incline.

In the 6300-ft. level, beginning at a point 180 ft. inside No. 1 incline and continuing for 170 ft., the timbers were knocked out, the bottom heaved and the low side rib was slightly pushed up, while for a length of 240 ft. inside this point the timbers were knocked out, the pavement was broken and the low side coal rib pushed uphill, closing the level. This low side rib slid up from the parting in the seam, leaving 14 in. of roof coal intact.

All the wood packs on the low side of the roadways in this section were demolished and some stone fell from the roof. Inside this damaged area the level was intact for a length of 60 ft. Inside this again for a distance of 200 ft. the low side rib under the roof coal was pushed uphill, the bottom was heaved up a height of 4 to 5 ft., timbers were knocked out and the high side packs were squeezed to about half their original height, without, however, any fall of roof. From this section to the face, a distance of 50 ft., the bottom was slightly heaved.

The 6500-ft. high level was filled with coal for a distance extending from No. 1 incline to the longwall face.

In the headway between the 6300-ft. level and the 6500-ft. counter level, 300 ft. outside the longwall face, the wood packs on the east side of the head were moved bodily about 2 ft. to the west and some coal was thrown off the east side rib.



SECTION AT 1

All east side chocks in this place have been moved about two feet westward. With the exception of three chocks at point of section the individual sticks have not been displaced. At the section shown above, the upper part of the chock, the booms and the collar boom which supports the booms between chocks have not moved but lower portion has moved as a unit.

SECTION AT 2

SECTION AT 3

SECTION AT 4

SECTION AT 5

SECTION AT 6

Fig. 3.—Sections of roadways at locations shown on Fig. 4.

Beginning at a point 170 ft. inside No. 1 incline on the 6500-ft. level the timber was knocked out, the coal was thrown from the high side, the bottom was broken and heaved and in places the low side pillar was pushed up for about 2 ft. A considerable quantity of stone fell in this area, which was 340 ft. long. Inside this there was an undamaged length of 50 ft. and beyond this for 190 ft. the bottom was heaved, the timbers were knocked out from the low side and the low side rib was pushed uphill. From this point to the face no damage was done.

In the 6500-ft. mine bord the pavement was cracked for a distance of 1475 ft. east of the fan slope. This crack extended along the low side prop line in most places but occasionally appeared in the center of the roadway. The high side rib of this mine bord threw off sufficient coal for a length of 120 ft., beginning 470 ft. from the longwall face, to practically fill the bord.

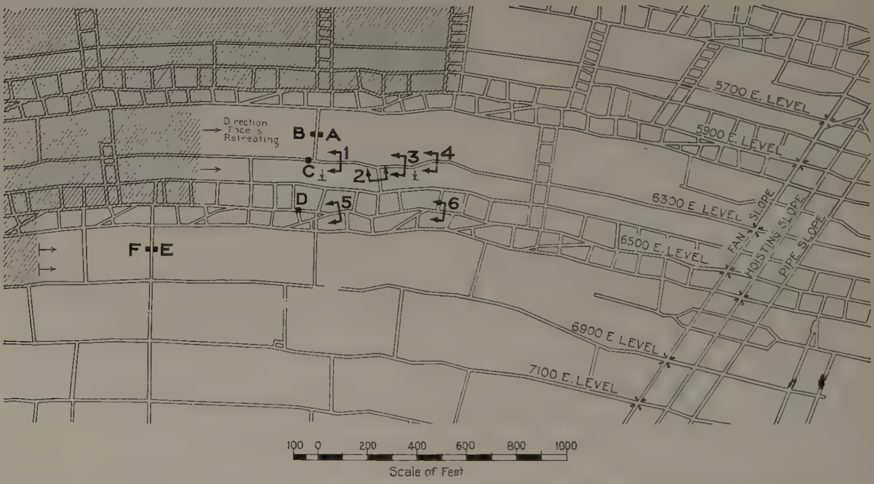


FIG. 4.—LOCATION OF MEASURING STATIONS AND OF SECTIONS OF ROADWAY (FIG. 3) BEFORE AND AFTER BUMP OF NOVEMBER 22, 1931.

The wind blast resulting from this last bump wrecked most of the doors and stoppings in the sections affected. In all cases the stoppings were either blown uphill or to the west, with the exception of the door in the 6500-ft. mine bord close to the longwall face. This door was blown to the east.

Indications Preceding Bumps

For a week, prior to Nov. 22, the coal on the 6500-ft. face had been tightening and had a tendency to be set. Other conditions were normal. Conditions on the 6300-ft. wall were normal. The longwall faces were not in any case affected by this bump. For several days after these bumps, working, grinding and thumping in the upper strata could be heard.

On Fig. 3 are shown cross-sections taken at the various points marked, for the sake of clarity, on the plan, Fig. 4. These indicate the condition of the roadways before and after the bump.

Previous experience, in the bumps of the year 1928, had shown that it was safe to continue working for a limited distance after a bump, therefore after the damage to the levels had been cleaned up these faces were advanced another 120 ft. without experiencing any further trouble and were then stopped.

Observation Stations

The complete extraction of coal seams apparently is always accompanied by a "weighting" of the roof immediately in advance of area of extraction, the extent of this action varying with the depth of mining, method of roof support and the nature of the roof, pavement and coal.

Halbaum, who was one of the first to draw attention to this action, coined the term "underdraw" as describing the distance ahead of the workings in which the coal is affected in this manner.[3]

Winstanley in his paper on roof control[4] cites what might appear to be an extreme case of this action taking place in the solid coal 70 ft. ahead of longwall faces. In this instance the height of the seam was 4 ft., the roof massive sandstone 60 ft. thick and the depth of cover 1250 feet.

Briggs and Ferguson in a comprehensive paper on the movements of rocks also instance this underdraw.[5]

It appeared to the writer that there might be some intimate connection between this advance-weighting and the disastrous bumps that had taken place on the 5300 and 5700-ft. levels and accordingly, late in the year 1930, it was decided to obtain more definite information regarding the extent of the underdraw and observation stations were established well in advance of the longwall faces in the following year.

Fig. 4 is a plan showing the positions of the observation stations *A*, *B*, *C*, *D*, *E*, and *F*, together with the relative positions of the longwall faces at the time these stations were established. Stations *A* and *B* were placed at No. 5 incline at the ends of narrow cuts driven about 8 ft. into the solid coal on either side of the incline. At the end of these cuts plugs were driven into holes bored in the roof and pavement and so placed that light iron rods freely suspended from the roof plugs hung vertically above the pavement plugs.

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[3] H. W. G. Halbaum: The Great Planes of Strain in the Absolute Roof of Mines. *Trans.* Inst. Min. Engrs. (1905–06) **30,** 175.

[4] A. Winstanley: Longwall Roof Control. *Trans.* Inst. Min. Engrs. (1930–31) **81,** 384.

[5] H. Briggs and W. Ferguson: Investigation of Mining Subsidence at Barbauchlaw Colliery, West Lothian. *Trans.* Inst. Min. Engrs. (1932–33) **85,** 313.

Stations *C* and *D* were established more or less in line with the above but to the dip and in this instance the plugs were placed in the centers of the 6300 and 6500-ft. levels. The convergence of the roof and pavement was ascertained by measuring the distance between the end of the rod and the point of a spike securely fixed in the pavement plug.

In this district both the roof and pavement are strong and, in the case of stations *A* and *B*, the narrow width of the cuts backed up by large pillars of coal would preclude the possibility of any bending of the roof or pavement in the cuts themselves seriously impairing the accuracy of the measurements. However, considerable bending of the roof and pavement did take place across the width of the incline itself, this action compressing and crushing the coal on the ribs of the incline, but as far as could be seen the backs of the cuts were not affected.

At stations *C* and *D* the width of the levels would certainly permit of a bending of the roof or spring of the pavement and consequently the measurements obtained at these stations can only be regarded as indicative of the action subsequently found to be taking place. These levels were driven 12 ft. wide, but, with the passage of time and continual spalling of coal from the high side rib, this width had steadily increased until, in certain parts, these levels, when measured next to the roof, showed a span of 25 feet.

In selecting the sites for these observation stations due consideration was given to placing them in what was considered to be undisturbed ground sufficiently far in advance of the longwall faces to be free from any danger of advance "weighting."

Measurements of the convergence of roof and pavement were taken at the observation stations at regular intervals and at the same time the distances of the longwall faces from these various stations were noted. These observations, while admittedly elementary in character, were the best that could be done under the circumstances because the colliery staff had not the time to take daily or even weekly precision levels of the elevations of the roof and pavement plugs.

The records of the observations are listed in Appendices 1 and 2 and are shown plotted on Figs. 5, 6 and 7, the vertical ordinates representing the distance of the station from the longwall face, while the horizontal ordinate represents in inches the approach of roof and pavement towards one another.

In this mine it was a well-known fact that the coal seam did flow under pressure, instances having been observed of roadways after a lapse of time being completely filled with coal. Although this action at stations *A* and *B* was not apparent from ordinary inspection, additional information was sought on this point at stations *E* and *F*. To this end plugs were driven into holes bored about mid-seam in the high and low side ribs and wires were stretched between these plugs. These wires were placed just making contact with the rods hanging from the roof plugs, consequently any departure of the wires from the rods would represent a movement or flow of the coal carrying with it the plugs holding the wire.

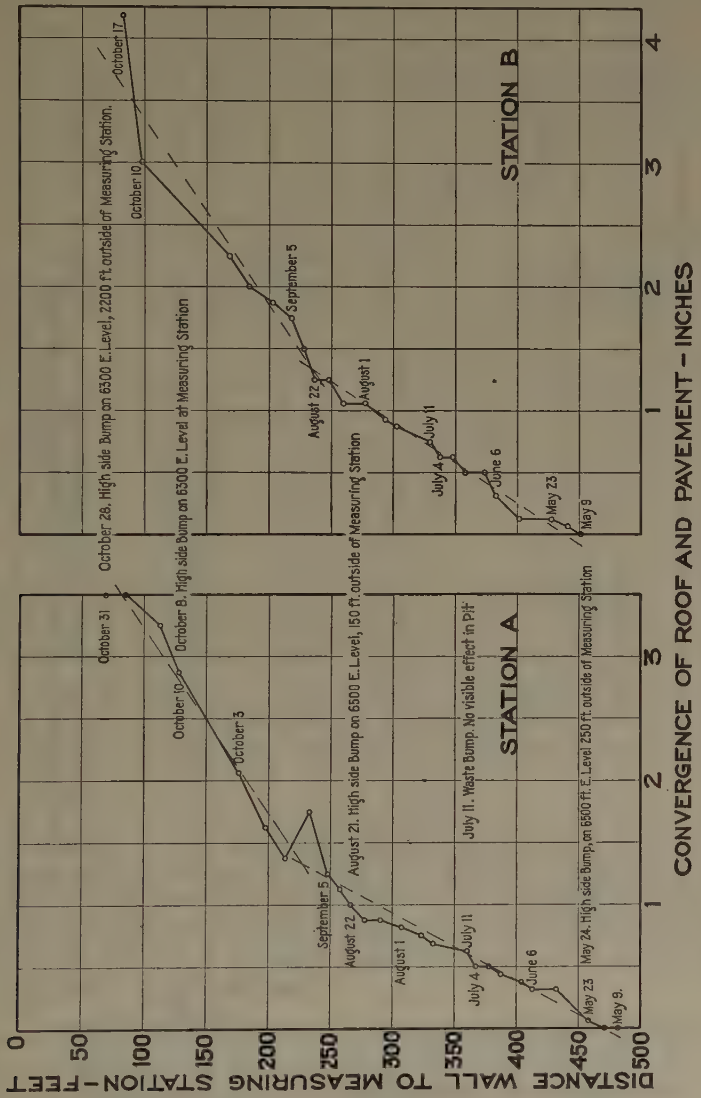

FIG. 5.—RECORD OF CONVERGENCE OF ROOF AND PAVEMENT, STATIONS A AND B.

ANALYSIS OF RECORDS FROM OBSERVATION STATIONS

When the second set of measurements was taken at stations *A*, *B*, *C* and *D* it was found that a distinct movement of convergence of roof and pavement was already in progress. This movement continued at a more or less uniform rate until, with the nearer approach of the longwall faces, the rate was greatly accelerated. On the charts of these movements, Figs. 5 and 6, the dates of all the known minor bumps in that district of the mine have also been recorded. The increased rate of convergence after a bump should be noted.

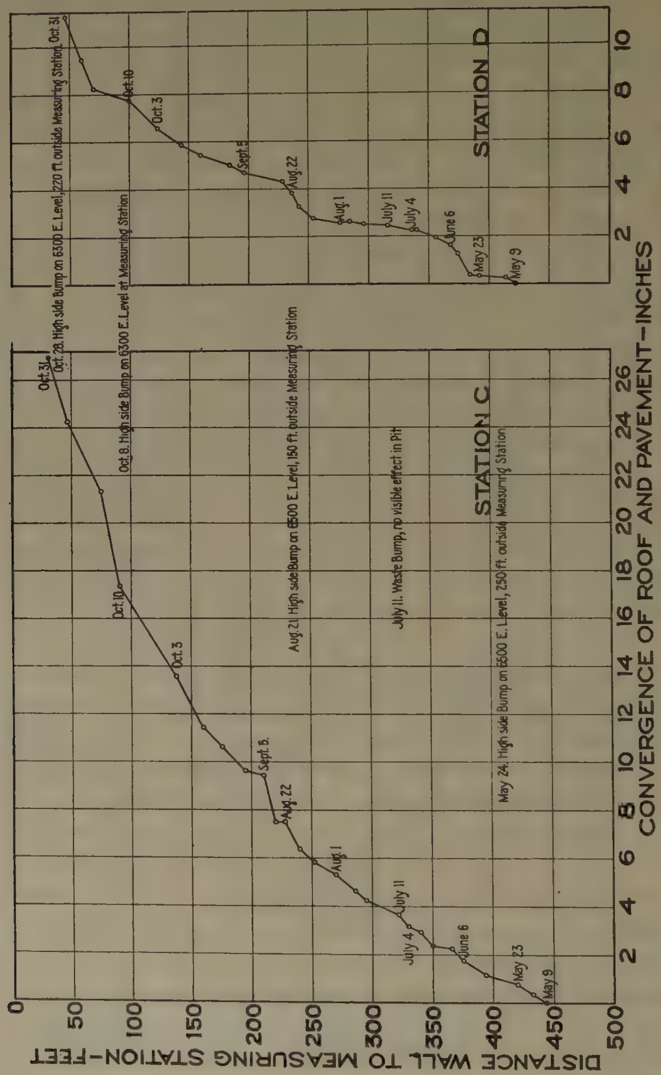

FIG. 6.—RECORD OF CONVERGENCE OF ROOF AND PAVEMENT, STATIONS C AND D.

On May 24, 1931, a high side bump affected the 6500-ft. level 250 ft. outside of the measuring station. Between the measuring dates of May 27 and June 6 the rate of roof settlement at station *D* on the 6500-ft. level was greatly accelerated until June 13, when the rate of settlement diminished.

On July 11 a heavy bump was heard most plainly on 6300-ft. east level but no damage was done to roadways.

On Aug. 21, on the 6500-ft. east level, five high side props were broken and about 21 tons of coal discharged from high side rib. Center of disturbance was 380 ft. in advance of the longwall face and 150 ft. outby from observation station *D*.

On Oct. 8, on the 6300-ft. east level, several high side props were knocked out and 12 tons of coal discharged from high side rib. Center of disturbance was 210 ft. ahead of longwall face and 120 ft. outby from station *C*.

On Oct. 28, on the 6300-ft. east level, miners were stripping loose coal off high side of level in order to put in wooden packs when a bump took place, discharging about eight tons of coal. Center of disturbance was 270 ft. in advance of longwall face and 220 ft. outby from station *C*.

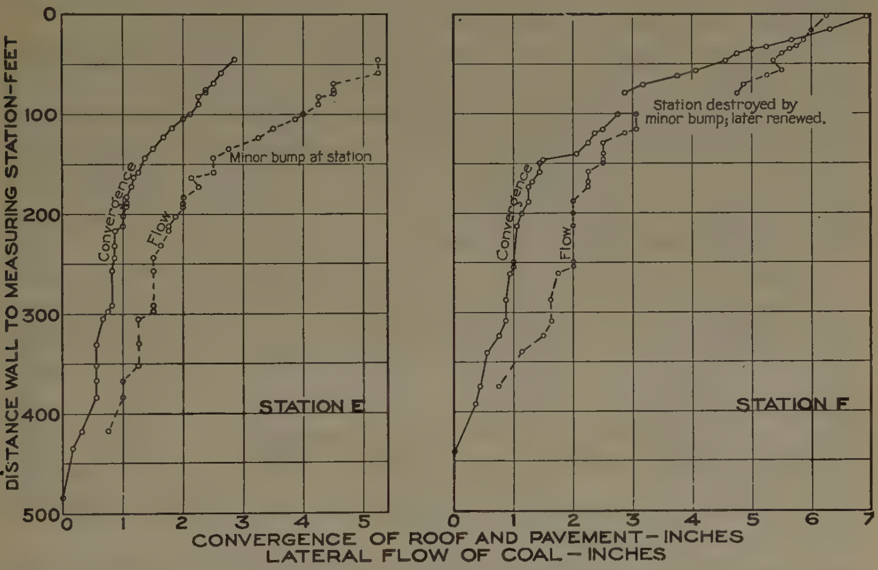

FIG. 7.—RECORD OF CONVERGENCE OF ROOF AND PAVEMENT AND OF LATERAL FLOW OF COAL, STATIONS E AND F.

On Oct. 31, on the 6300-ft. east level, about 150 ft. of the level was affected, and a considerable amount of coal was discharged from high side rib. Center of the disturbance was 680 ft. in advance of longwall face and 715 ft. from station *C*.

On Nov. 7, on the 6500-ft. east level, 140 ft. of level was affected starting from a point 60 ft. outside of the longwall face. Approximately 250 tons of coal was violently discharged from high side rib and the roadway was thrown from the high side of the level to the low side.

The measurements at station *A* were continued until it was wrecked by a slight bump when the wall was still 69 ft. distant. Records at station *B* were obtained until the longwall face was only 83 ft. distant, when a slight bump completely filled the cut with loose coal and generally wrecked it.

The data on these charts indicate that the complete extraction of coal on the longwall faces is accompanied by a gradual increase of stress on the pillars ahead of the longwall faces followed by a partial collapse of the pillar offering the resistance opposing such stress leading to a redistribution of forces. Undoubtedly the minor high side bumps are the manifestations of the sudden collapse of this resistance. It also appears from these charts that there is a time lag between the occurrence of a minor bump and the increased rate of convergence at the station, this lag varying with the intensity of the bump and the distance from the station.

It is evident therefore that the adjustment of stresses, after one of these minor bumps, is accomplished gradually, probably taking place in a series of jerks, and while this readjustment is in process certain pillars in the area affected must again be stressed unduly, such an action further weakening the already impaired pillars ahead of the walls.

The records obtained from stations *E* and *F*, Fig. 7 and appendix 2, prove that the convergences noted are attributable in part to a flow of the coal. At station *E*, after the convergence was 0.875 in., the wire had departed 1.75 in. in a lateral direction from the measuring rod, the flow of coal being in the direction of the relief of pressure or towards the open incline, the final measurements being a departure of 5.25 in. for a total convergence of roof and pavement of 2.8675 inches.

Similarly at station *F*, on the opposite side of the incline from station *E*, the total departure of the wire, again in the direction of relief of pressure or towards the incline, was 6.25 in. for a total convergence of 6.9375 inches. In this instance the convergence of roof and pavement showed more movement than at station *E*, owing to the fact that this station was placed on the east side of the incline, and therefore in a pillar steadily decreasing in size with the advance of the longwall face, whereas station *E* had a large body of coal at its back to carry the weight which crushed the diminishing pillar at station *F*.

It is interesting to observe from the charts that the flow of coal is not a continuous action but takes place in a series of steps, indicating that a movement of the coal temporarily relieves the overlying weight necessitating a readjustment of pressure-back in the pillar before sufficient weight can be accumulated to move the coal forward again.

If an attempt is made to rectify the curve by two straight lines, as shown dotted on Fig. 5, it will be noticed that these two lines have greatly different slopes, indicating a change in the rate of convergence. The steeper line represents the flow of the coal due to the great weight of the superincumbent strata while the more nearly flat line represents accelerated convergence, which may be taken as due to the superincumbent weight plus that referred to as "advance-weighting" or "underdraw" brought into play by the approach of the working face. There is yet another view that may be taken of the chart—that the intersection point of the two lines referred to indicates the "yield point" of the coal and that when the waste approaches to within the indicated distance the pressure on the coal has so greatly increased that the elasticity or resilience has been destroyed and consequently the flow, and its inferential convergence of roof and pavement, becomes more rapid. Both views are complementary.

It is only reasonable to assume that the thinning of the seam due to flow attains its maximum at the coal face or rib and gradually diminishes to nothing somewhere back in the pillar. If it can be assumed that compression of the coal is not also an accompanying factor and that the thinning of the seam diminished evenly, a simple calculation would show that this thinning of the seam must extend for a distance of at least 30 to 35 ft. into the pillar. However, as compression of the seam is a factor that cannot be overlooked, it is probable that the extent of this thinning is greater than the calculated distance.

Reverting to the chart of station *A*, Fig. 5, the convergence of roof and pavement continued at a fairly uniform rate until the longwall face was within 275 ft., after which the rate of convergence suddenly increased, the change being attributable to advance-weighting from the longwall face, or, in other words, the station was now in the zone of the "underdraw." At station *B* this distance was 240 ft. and similarly at stations *C* and *D* the distances averaged 250 feet.

Where the ground had not been already unduly disturbed by low side bumps, the area of disturbances in these bumps centered about 240 ft. ahead of the longwall faces, and it is of more than passing interest to note that this distance is practically identical with the forward reach of the under-draw. At stations *E* and *F* the nature of the ground differs from that at the other stations, the floor being decidedly weaker, and, judging from the charts, the zone of the under-draw in this case did not extend more than 170 ft. ahead of the longwall face. From this it would appear that, other factors being equal, the zone of the under-draw depends upon the strength of the roof and pavement—the greater the strength, the greater the draw.

Winstanley[6] cites instances of measuring "rises in the roof just ahead of where subsidence began to increase rapidly in front of the face." The graph of the convergence of roof and pavement at station *A*, Fig. 5, shows clearly a divergence of roof and pavement amounting to 0.38 in. when the wall was 233 to 214 ft. distant. From additional observations taken at this time, reference to which will be made later on, this divergence was due to a re-elevation of the roof.

At station *D*, although this is not so clear on the graph, there was a divergence noted of 0.06 in. when the wall was from 284 to 276 ft. distant. At the other stations actual divergences were not noted, although this may have occurred, as the records show arrested convergence at station *B* when the wall was 193 to 184 ft. distant. Similarly at stations *C*, *E* and *F* the arrested convergences occurred when the walls were 228 to 220 ft., 193 to 184 and 201 to 189 ft. distant from the walls respectively.

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[6] Reference of footnote 4.

CAUSE OF BUMPS

There is such a remarkable similarity in the configurations of the workings in the bumps of April 2, July 3 and Oct. 12, 1928 and Nov. 22, 1931 that this similarity appears to be more than a mere coincidence. These workings had originally been developed, by the three-entry system—of which Fig. 8 is a typical layout—for retreating room and pillar work, but subsequently were adapted for longwall retreating by driving a halfway level. In this part of the mine the roof and floor immediately next the coal are strong and normally do not yield.

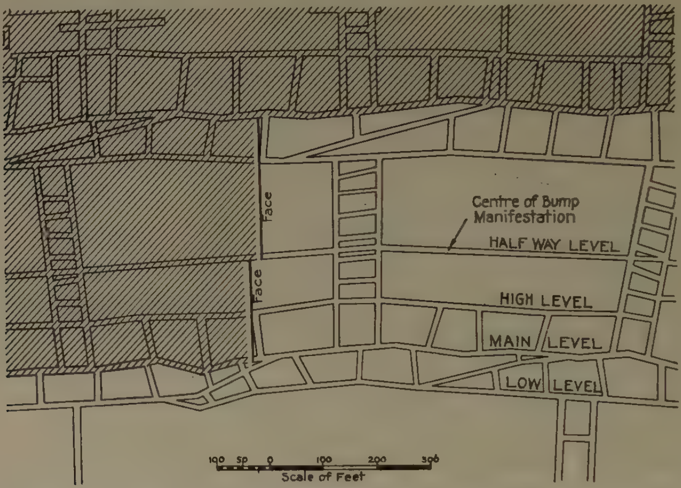

FIG. 8.—TYPICAL LAYOUT OF WORKINGS IN WHICH SEVERE BUMPS HAVE OCCURRED.

The roadways shown on the plan were driven 12 ft. wide in the first instance, but with the passage of time and assistance of gravity on a pitch of 20°, continued spalling from the high side has widened out these roadways to 25 ft. in places. Beyond this point weight has undoubtedly loosened the coal for a considerable distance. The coal on the low side rib of a roadway, although loose, does not spall off, gravity in this case tending to prevent this action, consequently the roadways do not widen in this direction. The pillars on the high and low sides of the levels were originally 90 ft. wide but with the conditions referred to above the effective width of a pillar is probably less than 60 ft. Crosscuts were driven on 97-ft. centers.

The sandstones at Springhill, where observed in cross-measure tunnels and in exposures on the surface, are strong, fine-grained, dense rocks and unusually free from cross jointing. Substantial beds of this nature are capable of spanning considerable intervals without breaking and, furthermore, will distribute the weight of the superincumbent strata over quite an extensive area. Consequently the weakened pillars referred to above have to carry the full burden of weight formerly borne by the full-sized pillars.

Take the case of the pillars on either side of the 6500-ft. level. The depth of cover at this point is, in round numbers, 2600 ft., a depth which, taking the average weight of rock at 160 lb. per cubic foot, impresses on the coal a weight of 2888 lb. per square inch.

The level, together with the pillars and roadways immediately to the dip and rise, measured center to center, constitute a width of some 200 ft. of which the first drivage removed 24 ft., leaving 178 ft. of pillars supporting the roof, making the vertical loading on these pillars—neglecting all lateral support—approximately 3750 lb. per sq. in. The spalling of these pillars, however, has reduced the effective support to a width of probably not more than 138 ft., thereby—again neglecting lateral support—increasing the vertical loading to some 5350 lb. per square inch.

This concentration of weight accompanied by advance weighting from the longwall extraction on to pillars ever growing smaller is held to be responsible for the minor high side bumps experienced on the roadways in advance of the longwall faces, this action continuing until all resistance is crushed out of the pillars on either side of the main level.

The pillar to the rise of the halfway level is of such substantial size that the coal in the center of such a pillar is virtually confined and therefore capable of offering enormous resistance to the pressure arising from the superincumbent load. The records of convergence at stations *A* and *B* appear to bear out this contention, the maximum convergence observed before the stations were finally destroyed being approximately 4 inches. The pillar to the low side of the halfway level, although in most instances not much wider than the pillar to the rise of the main level, nevertheless, owing to the absence of crosscuts, is capable of offering resistance to pressure after the resistance has been crushed out of the pillars on either side of the main level.

The greater convergence observed at station *C* on the halfway level than at station *D* on the main level, 26.93 in. as compared with 11.25 in., would appear to offer evidence controverting the above statement. It might be claimed that the smaller amount of convergence at station *D* was due to the overlying sandstone bed projecting like a beam—fixed at one end—from the large pillar below the low level, carrying the superimposed weight and thus preventing the pillars it protected from carrying the full weight. However, if this were true for the main level area it should be equally true for the halfway level area, as the pillar to the rise of the halfway level is of very substantial size, but as evidence does not bear this out, this theory may be dismissed.

Other observations made at these two stations have not been recorded in this paper, as there are doubts about the accuracy of the methods employed, but some of these are now given to show the relative actions.

When station *C* was established in the halfway level on May 9, 1931, the distance from the rod hanging from the roof to a mark on the low side rib was 8 ft. 4¾ in. On Aug. 1 this distance was 8 ft. 3 in. and on Oct. 24 the final measurement was 7 ft. 8½ in., the indications being that the low side rib had moved 8¼ in. uphill.

At station *D* on the main level the total of a similar movement over the whole of this period was only ½ inch.

Attempts were also made to determine individual roof and pavement movements. At station *C* the total apparent upward movement of the pavement was 12⁹⁄₁₆ in., while at station *D* no such movement was shown. Again, at station *C* the lateral displacement of the roof and pavement plugs in a direction parallel to the level was 4 in., the pavement appearing to have moved in an easterly direction or towards the longwall faces. At station *D* the roof plug remained vertically above the pavement plug until the station was destroyed.

These observations show that the ground at station *D* was apparently subject to no great pressure, while the ground at station *C* being on the move all the time, indicated that heavy pressure was being exerted at that point. The writer is of the opinion that a very considerable amount of convergence had already taken place before station *D* was established and that, furthermore, most of the powers of resistance having been crushed out of the pillars on either side of the level, these pillars were unable to transmit the full pressure and that consequently the pavement reaction was correspondingly less.

The overlying weight normally on these pillars would in these circumstances be carried by the overlying sandstone bed bridging across the level from the solid coal below the low level to the pillar below the halfway level. This latter pillar, then acting as one of the abutments of the bridge, would be stressed excessively and the pavement of the halfway level in reacting to this pressure would certainly spring. A weaker pavement would most certainly have broken under this pressure.

It would not, therefore, appear to be unreasonable to suppose that when the pressure of this bridging effect combined with advance weighting of the underdraw reaches a certain maximum, the pillar on the low side of the halfway level is no longer able to resist and gives way suddenly along the lines of least resistance, i. e. up and downhill, giving rise to these phenomena of low side bumps in which the seam as a whole is pushed bodily in either direction.

It has been shown that the flowage of coal takes place in steps or jerks, and one would expect that the redistribution of pressure which must take place as soon as the coal begins to flow would also be of a similar jerky nature, and it might very well be that such a jerk is the necessary culminating effort required to disrupt the pillar.

It is quite possible, although it would not appear to be necessary in this type of bump, that the culminating cause may be a jar or rearrangement of stresses due to the sudden failure of a cantilever of sandstone overhanging the waste but, if such is the case the wastes, as far as can be seen, give no indication of any unusual movement having taken place.

On the other hand, the jar or tremor might arise from the collapse of a large arch span in the upper roof. This theory, although not held accountable for the type of low side bump under discussion, is undoubtedly the cause of the district bumps and the readjustment of stresses after one of these bumps is plainly indicated by the sounds of the roof working and grinding for several days after a bump. With a low side bump these indications of readjustment are not nearly so prolonged.

The fact that the manifestations of these low side bumps always occur some distance in advance of the longwall faces, and that the faces themselves are never damaged, is accounted for by the fact that advance-weighting has crushed the coal for this distance and has destroyed the powers of resistance necessary to produce the requisite conditions.

Our observations had shown that the effects of the advance-weighting had in all probability shattered the pillar for a distance of some 250 ft. ahead, and consequently it was not thought that stresses sufficiently great to induce another bump could be set up in a further advance of, say, half that distance. Accordingly it was considered safe to advance the longwall faces of the 6300 and 6500-ft. levels for another 120 ft. after the severe low side bump of November, 1931. Whether this inference was correct or not, the fact remains that this distance was covered without any more disturbances.

Again, if these theories as to the cause of low side bumps are correct, it should be possible by artificial means to upset the delicate balance existing between pressure and resistance, and the use of explosives is suggested for this purpose. This could be accomplished by boring long holes into the pillars in advance of the longwall faces, springing these holes separately by initial charges of explosives and then using heavier charges fired simultaneously from a safe vantage point. If the opposing stresses are closely balanced, the loosening and jarring effect of the discharge should be capable of upsetting this balance and inducing a bump.

Unfortunately, if this were accomplished, although this particular hazard would be removed, there would still be the cost of reclaiming the roadways coupled with the loss due to temporarily reduced output; consequently, until coal is more valuable the idea is of academic rather than practical importance.

If the foregoing theory as to the cause of low side bumps can be accepted as correct, it follows that the method of development has been responsible for producing conditions whereby bumps can occur and, carrying this argument further, it should also follow that by using the single entry system for development, the size of the pillars would be such that they could withstand the pressure and bumps should not occur.

The 6900-ft. east level and all levels below this, were developed by the single entry system, and the longwall face on the 6900-ft. level was worked without any undue incidents, up to a point 150 ft. short of the positions of longwall faces on the 6300 and 6500-ft. levels at the time of the bump in November, 1931.

The decision to stop this face was reached only after very thorough deliberations, the overriding consideration being safety, as it was considered that further advance of the face into this hitherto hazardous zone might be the means of inducing another bump and thereby possibly causing loss of life.

The writer does not see any way of preventing district bumps except by the possible means of solid packing of all wastes but the cost of so doing under present conditions would be prohibitive. Even this precaution might fail in its entirety, no method of packing completely eliminating subsidence, but it should certainly succeed in reducing the severity of such bumps.

A list of all known bumps that have occurred since Mr. Herd's paper was written are in Appendix 3.

ACKNOWLEDGMENTS

The writer would like to thank Mr. A. K. McLeod, District Superintendent, Mr. H. C. M. Gordon, Assistant Mining Engineer and Mr. H. Peppard for their assistance in supplying information, recording measurements at observation stations, etc., and Mr. S. C. Mifflen, Office Engineer, for reading the proofs and making certain valuable suggestions.

DISCUSSION

(George S. Rice presiding)

G. S. RICE,* Washington, D. C. (written discussion).—The Committee is indebted to Mr. McCall for following up the study begun by the late Colonel Herd,[1] in describing further remarkable experiences in the Springhill mine from bumps subsequent to certain changes in mining methods. Also, for his special studies of convergence of roof and floor prior to and following bumps, after the method of testing convergence in mine workings developed by the Safety in Mines Research Board of Great Britain.

Naturally I feel great interest in the general subject, having made recommendations for changing the original method of working by room and pillar methods, in a report to the Mines Department of Nova Scotia in 1924, to a method of retreating longwall, which were agreed to in principle by the mining company.

The continuance of bumps, though less frequent and possibly less severe since the change in method of mining, is regrettable, but I ascribe some of these bumps in

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* Chief Mining Engineer, U. S. Bureau of Mines.

[1] Reference of footnote 1.

APPENDIX 1.—Observations of Convergence

Date, 1931	Station A		Station B		Station C		Station D	
	Distance Wall to Station, Ft.	Progressive Convergence, In.	Distance Wall to Station, Ft.	Progressive Convergence, In.	Distance Wall to Station, Ft.	Progressive Convergence, In.	Distance Wall to Station, Ft.	Progressive Convergence, In.
May 9	482		452		444		422	
16	471		441	0.0625	433	0.3750	414	0.25
23	458	0.0625	428	0.1250	420	0.7500	392	0.3125
27	432	0.3125	402	0.1250	394	1.1250	384	0.3750
June 6	413	0.3125	383	0.3125	375	1.7500	374	1.25
13	404	0.3750	374	0.5000	366	2.2500	368	1.625
19	388	0.4375	358	0.5000	350	2.3750	356	1.9375
27	378	0.5000	348	0.6250	340	2.9375	340	2.25
July 4	368	0.5000	338	0.6250	330	3.1875	336	2.25
11	360	0.6250	330	0.7500	322	3.6875	316	2.4375
18	333	0.6875	303	0.8750	395	4.2500	296	2.50
24	324	0.7500	294	0.9375	286	4.6250	284	2.625
Aug. 1	308	0.8125	278	1.0625	270	5.3125	276	2.5625
8	290	0.8750	260	1.0625	252	5.8125	254	2.75
15	278	0.8750	248	1.2500	240	6.3750	242	3.25
22	266	1.0000	236	1.2500	228	7.5000	236	3.8125
29	258	1.1250	228	1.5000	220	7.5000	228	4.3125
Sept. 5	248	1.2500	218	1.7500	210	9.4375	196	4.6875
12	233	1.7500	203	1.8750	195	9.6250	184	5.0
19	214	1.3700	184	2.0000	176	10.6250	160	5.4375
26	198	1.6250	168	2.2500	160	11.4375	144	5.875
Oct. 3	176	2.0625			138	13.5625	124	6.5625
10	128	2.8750	98	3.0000	90	17.3750	100	7.75
17	113	3.2500	83	4.1875	75	21.3125	70	8.25
24	85	3.5000	Station destroyed by bump.		47	24.2500	60	9.4375
31	69	3.5000			31	26.9375	46	11.25
	Station destroyed by bump.				Station destroyed by bump.		Station destroyed by bump.	

DISCUSSION (*Continued*)

part to the nearness of the retreating faces to the not fully settled former room and pillar workings, and in part to the individual retreating faces having long steps between them, instead of short steps making practically one long face, which for roof support is always desirable in longwall work, either in advancing or retreating.

These steps at Springhill appear to be as long as individual faces, an arrangement found equally bad in advancing longwall where there is strong roof. The projecting squares of coal receiving no support from the old goave on the work-out rise side, for the support of an exceedingly heavy roof, probably cause its weight to be concentrated on the projecting corner of coal just as on a pillar.

This concentration of pressure may be partly the cause of convergence of roof and floor recorded by Mr. McCall and in turn may be one of the causes of continuance of occurrence of bumps. My tentative hypothesis is that the immediate roof—that is, the section extending up to the massive sandstone stratum known to exist above—may have sufficient weight to cause the recorded convergence, but the sandstone stratum, as was indicated in the past, is very strong and rigid and until broken cantilevers from over the unmined area to where it is broken or sheared in the goave, behind the longwall faces. Finally, however, when the tensile strength of the sandstone stratum in the upper part of the hypothetical cantilevering slab is exceeded and

APPENDIX 2.—Observation of Convergence, Station E

Date, 1931	Distance Wall to Station, Ft.	Measurement Roof to Pavement, Ft.	Progressive Convergence, In.	Progressive Lateral Movement of Wire, In.
Jan. 4	484	9.7135		
20	435	9.6979	0.1875	
Feb. 2	418	9.6875	0.3125	0.7500
17	384	9.6667	0.5625	1.0000
27	367	9.6667	0.5625	1.0000
Mar. 5	352	9.6667	0.5625	1.2500
14	331	9.6667	0.5625	1.2500
21	305	9.6562	0.6875	1.2500
29	298	9.6510	0.7500	1.5000
Apr. 6	292	9.6458	0.8125	1.5000
13	257	9.6458	0.8125	1.5000
20	244	9.6406	0.8750	1.5000
28	232	9.6406	0.8750	1.6250
May 9	217	9.6406	0.8750	1.7500
19	212	9.6302	1.0000	1.7500
27	202	9.6302	1.0000	1.8750
June 2	193	9.6250	1.0625	2.0000
9	190	9.6250	1.0625	2.0000
16	184	9.6250	1.0625	2.0000
24	173	9.6198	1.1250	2.2500
30	163	9.6146	1.1875	2.1250
July 7	159	9.6094	1.2500	2.5000
14	144	9.5990	1.3750	2.5000
22	135	9.5885	1.5000	2.7500
28	123	9.5729	1.6875	3.2500
Aug. 4	114	9.5625	1.8125	3.5000
12	105	9.5469	2.0000	3.8750
18	100	9.5365	2.1250	4.0000
26	90	9.5260	2.2500	4.2500
Sept. 1	83	9.5260	2.2500	4.2500
8	78	9.5156	2.3750	4.5000
15	76	9.5156	2.3750	4.5000
22	69	9.5052	2.5000	4.5000
30	59	9.4948	2.6250	5.2500
Oct. 7	45	9.4896	2.8675	5.2500

DISCUSSION (*Continued*)

it ruptures, it produces what I have previously termed a "shock bump," in distinction to a "pressure bump" when the strength of a coal pillar is exceeded, such as experienced in eastern Kentucky mines and elsewhere. The breaking and striking of a section of a massive stratum imparts a hammerlike blow on the coal at some point that previously supported a heavy unit load, and crushes the coal locally or, where the coal is stronger, transfers the blow to the floor.

The arch theory in cases where pressure bumps are experienced I have advanced in papers for many years, but I cannot agree with Mr. R. Dawson Hall that the practical evidence of arch stresses is indicated at Springhill. It does not seem to

APPENDIX 3.—Bumps in No. 2 Mine, Springhill, Nova Scotia, Jan. 22, 1929 to June 27, 1933

No.	Date	Location	Remarks
293	1929 Jan. 22	At face of new slope, 70 ft. above 6900-ft. west level.	Five sets of timber knocked out and some roof stone and coal down.
294	Feb. 11	High side of 6500-ft. east level between main slope and fanway.	Twelve high side props and one set of timber knocked out; ten boxes of coal and two of stone down. Two men slightly injured.
295	Feb. 27	Bottom turnout, 6900-ft. east level.	Ten boxes of coal knocked down.
296	Feb. 27	Main slope between 5700 and 5900-ft. levels.	Five props broken.
297	Feb. 27	6500-ft. east level, bottom turnout along high side of level and up east side of first head inside fanway.	Ten boxes of coal down from high side of level and from the head inside of fanway.
298	Feb. 27	300 ft. from slope on bottom turnout, 6500-ft. east level.	Bump on high side of level knocked out four steel booms, ten boxes of stone and five of coal.
299	Mar. 14	100 ft. outside No. 10 incline 6300-ft. east level.	Four boxes of coal from high side rib.
300	Mar. 29	From main slope to pipe slope on 6900-ft. west level.	Fifteen boxes of coal, five boxes stone down.
301	Apr. 4	On the 6300-ft. east level, 300 ft. outside No. 6 incline.	High side bump. Twelve boxes of coal down; four props broken.
302	May 8	25 ft. from main slope on high side, 7400-ft. east level.	Two props and ten boxes of coal knocked from high side rib.
303	May 9	On 6300-ft. east level, 150 ft. from main slope.	One set of timber knocked out; two boxes of stone and six boxes of coal down.
304	Dec. 10	At intersection of 6500-ft. level and slope.	Three high side props and one boom knocked out. Three boxes of coal down.
305	1930 Feb. 28	At longwall face on 6300-ft. east level, face being 25 ft. outside No. 8 incline.	Twenty-five boxes of coal down along face.
306	Mar. 25	At face of level, 8600-ft. east level, west sinking 620 ft. from sinking.	Ten boxes of coal off high side rib.
307	Apr. 10	At face of No. 5 incline, which was 60 ft. above 6300-ft. east level.	Man's arm broken. Some stone came down, knocked out one prop and broke several stringers.
308	June 7	East rib, main slope between 6500-ft. level and mine bord.	Four sets of timber knocked out and six boxes of coal down.
309	July 19	6500-ft. east level, inside No. 3 slant.	Props and booms knocked out for 64 ft. and 120 boxes coal down, but not much stone.
310	Aug. 22	Main slope above 6500 lodgment.	One prop knocked out and about three boxes of coal down.
311	Oct. 29	6500-ft. east longwall face, 30 ft. above counter level, 400 ft. inside No. 6 incline.	Thirty boxes of stone down along face, breaking timber.
312	Oct. 21	At bottom end of pans, 6300-ft. east longwall face, 200 ft. inside No. 6 incline.	Knocked out high side pack, broke four low side props and fourteen boxes stone down.
313	Dec. 23	Main slope 5800 to 5900-ft. levels.	Three boxes coal from east rib. Four props displaced, six broken.
314	Dec. 26	6500-ft. level east.	Several props and booms broken. Ten boxes of coal down.
315	1931 Feb. 6	6300-ft. level east.	No damage, small quantity of coal scattered along level.
316	Feb. 15	8600-ft. level east high side rib 1400 ft. from slope.	Thirteen boxes of coal off high side rib. One prop knocked out.
317	Mar. 4	6900-ft. level east 30 ft. from slope.	Knocked out five high side props and five boxes coal.
318	Mar. 31	Pipe slope 150 ft. above 3300-ft. level on west rib.	Nine props out on west side. Eight boxes of coal and stone down.
319	Mar. 31	6300-ft. level east on level 1500 ft. from slope.	Six boxes of coal and one of stone down. High side bump.
320	Apr. 3	6300-ft. level east 300 ft. from slope.	Four high side props broken. Six boxes of coal of high side rib.
321	Apr. 13	6300-ft. level east 1800 ft. from slope.	Knocked out two and broke six high side props, ten boxes of coal off rib.
322	Apr. 17	Main slope 40 ft. above 6800-ft. pump on east rib.	Knocked out four props, three boxes of coal.
323	Apr. 17	6300-ft. level east 1400 ft. from slope.	Small amount of coal scattered along level.
324	Apr. 23	6300-ft. level east 1700 ft. from slope.	Three sets of timber down, ten boxes of coal. High side bump.
325	May 24	6500-ft. level east 1600 ft. from slope.	Ten boxes coal and ten boxes stone down, about 60 ft. of level damaged.
326	June 24	Main slope on east rib from 6100-ft. pump up 50 ft.	Shattered east rib for 50 ft., two booms, one prop down.

APPENDIX 3.—(*Continued*)

No.	Date	Location	Remarks
327	July 12		A heavy report seemed to be all over pit. No damage.
328	July 18	6500-ft. level 100 ft. from slope.	Several high side props and two booms down, five boxes of coal.
329	Aug. 21	6500 ft. level 1700 ft. from slope.	Twenty boxes of coal down, five high side props broken.
330	Oct. 8	6300 ft. level east.	Twelve boxes of coal down, several high side props and two booms down.
331	Oct. 28	6300-ft. level east 1700 ft. from slope.	High side bump, several boxes of coal fell. No damage.
332	Oct. 31	6300-ft. level east 1200 to 1350 ft. from slope.	Quantity of coal down; loaded eight boxes.
333	Nov. 7	6500-ft. level east Bull Wheel turnout.	About 100 ft. of level closed, high side bump, about 250 boxes of coal down.
334	Nov. 17	6700-ft. lodgment.	A few props out on fanway just above lodgment, one prop, two boxes coal down on main slope.
335	Nov. 22	6300-ft. level east, 6500-ft. counter level. 6500-ft. level 100 ft. inside No. 1 incline.	Falls of coal and stone roof. Levels badly damaged.
336	Nov. 22	6500-ft. level from No. 1 slant to slope. 6500-ft. mine bord No. 1 slant to slope. 6500-ft. counter level No. 1 incline to slope, 6900-ft. level from slope in 600 ft. fanway slope and pipe slope from 6500-ft. counter level to 7100-ft. level.	Falls of stone and coal, timber broken and knocked out. High side and east side bump.
337	Nov. 25	Main slope 25 ft. below 6500-ft. mine bord.	Two sets of timber and seven boxes of coal down.
338	Nov. 30	Main slope 30 ft. below 6100-ft. lodgment.	Knocked out two props, three boxes of coal.
	1932		
339	Jan. 13	Downhill head off 7400-ft. level.	Three sets of timber out at face, five boxes of coal down.
340	Feb. 3	30 ft. below 5400-ft. level on main slope.	One prop out, small quantity of coal down.
341	Feb. 9	6300-ft. longwall face.	Some stone down and two packs thrown out.
342	Feb. 9	6300-ft. level east 150 ft. outside No. 1 incline.	Some coal down from high side.
343	Feb. 16	Main slope 100 ft. above 5400-ft. level.	One box coal off east rib main slope; three boxes off east rib fanway.
344	Apr. 18	7800-ft. level, 350 ft. from auxiliary slope.	One set of timber, three props and twelve boxes of coal down.
345	May 5	7800-ft. level, on bottom turnout.	Twelve boxes coal thrown from high rib. No timber down.
346	June 26	6500-ft. level east from back head, No. 1 incline on to first head up.	All high side props out; booms and stone down; level closed.
347	July 25	On auxiliary slope, between 8300 and 8600-ft. levels.	Coal down along slope.
348	Aug. 21	7400-ft. level halfway between first head up and main slope.	Three boxes of coal off high side.
349	Nov. 16	7800-ft. longwall face.	Damage slight; knocked a little coal from ribs.
350	Nov. 25	6500-ft. mine bord, 300 ft. inside longwall face.	A few high side props broken.
	1933		
351	Jan. 18	7800-ft. level.	Three sets of timber out; coal off high side.
352	Mar. 2	Main slope and pipe slope or slope 100 ft. above electric pump, or pipe slope 40 ft. above pump head.	On slope six props and eight boxes of coal off east rib; on pipe slope five props and six boxes of coal off east rib.
353	Mar. 28	6900-ft. level east 75 ft. outside last head up.	Low side props knocked out, two boxes coal off low side rib, high side rib shattered.
354	Apr. 28	New head driving from 6900-ft. level to 6500-ft. mine bord.	Knocked out three sets timber; two boxes stone down.
355	June 1	6500-ft. level west, 250 ft. west of main slope for 30 ft.	Eleven low side props out; two booms down; six boxes coal off low side.
356	June 12	Main slope 20 ft. above 7400-ft. level for 40 ft. along east rib.	Knocked out three booms, six props, twelve boxes coal; three boxes stone down.
357	June 27	8600-ft. level, 800 ft. from auxiliary slope.	Six boxes coal down; two props broken.

apply when dealing with a thick strong sandstone or conglomerate stratum overlying a coal bed at some distance above it.

R. D. Hall,* New York, N. Y. (written discussion).—Mr. McCall's article calls attention to the fact that the incidence of pressure in large rock spans is not at the edge of the pillar but often some hundreds of feet beyond it. In my discussion, I shall use the expression "stress arch" to describe the resultant line of pressures by which the rock is sustained, whether the roof has formed itself into an arch or is simply an unbroken slab of rock, unchanged by falls within the undermined area. In fact, in Fig. 9, I have shown the rock as if it had not caved at all, though it probably has caved for some hundreds of feet. Had caves occurred, the stress arch would have greater vaulting, for the stress arch rises when interior falls occur that raise the middle third. The top of the resistant material under the surface will be termed the "crown" of the arch and the underside of the resistant rock will be termed the "soffit." As stated, the underside may be flat or curved. It is usually curved, but is shown flat in the illustration.

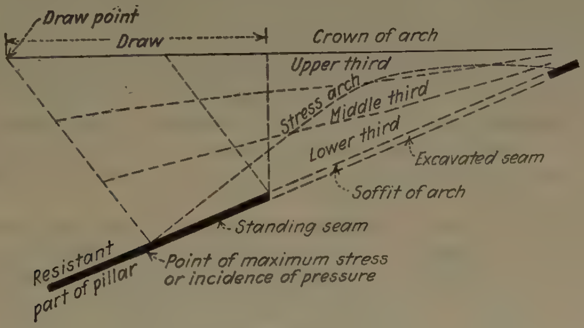

Fig. 9.

Where the stress line of the arch is above the middle third of the depth of the arch at any point, there is tension on the soffit at a point at right angles to the stress curve. Where the stress line is below the middle third of the depth of the arch at any point, tension is manifested on the crown at a point at right angles to the stress curve. As rock is much less resistant to tension than to compression, the rock gives way readily to tension, and the arch tries to drop, but any such descent is resisted by increased compressions which increase the compressive forces causing the resultants of compression and vertical pressure or weight to become flatter. Thus the stress arch flattens out and adjusts itself to a line nearer the middle third—a line that makes the arch give its maximum resistance to destructive forces. In designing an unreinforced arch for bridges, conduits and the like, the constructor arranges that the compression at the top of the stress arch shall be sufficient to keep it within the middle third of the voussoirs without bringing on them any more compression than they will stand with a sufficient factor of safety. He knows that if the stress keeps in the middle third and the voussoirs can withstand the pressures, there will be no tension and no collapse. But in the roof arch, as there are no voussoirs held only by cement, a little tension is permissible; so the roof stress arch is probably a little above the middle third at the top of the stress arch and a little below it at the edge of the pillar.

After the stress arch passes the edge of the pillar, it may be supported if the underclay rises and holds the coal against the roof. Under that condition, and only that, it will receive no new components of vertical stress. It will then no longer curve but become a tangent to the stress arch curve at that point. With 2000 ft. of rock cover, the tangent commences at somewhat less than 667 ft., above the edge of the coal pillar. It is easy to see that even with a level bed of coal it falls many times 100 ft. away from the pillar edge unless the span of the arch is short. With a dipping bed, as at Springhill, it may well be even further.

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* Engineering Editor, *Coal Age.*

In Springhill No. 2, the pillar edge, according to Fig. 8, is at the 5900-ft. east level, and the line of incidence of the pressure from the main arch had traveled on Nov. 22, 1931 to the 6500-ft. east level, though near the longwall naturally the incidence was apparently beyond that, for the low-side posts pushed out at the foot. It seems clear that near No. 1 incline the line of incidence is 600 ft. or more from the edge of pillar. The pillars, of course, above the 6500-ft. east level have given way more or less, but perhaps it is not the weakening of these pillars so much as the flattening of the stress arch from increased compression that has made the line of incidence travel from its former position between the levels 5700 and 5900 ft. Note that near No. 1 incline the edge of the pillar has not been advanced at all. The only work has been "longwall retreating" on the flanks of the levels.

Danger to miners seems solely in reaching and leaving working places. The haulage seems the only part of the working in jeopardy. Consequently, one is disposed to put the question: Should not the roadways used be those considerably to the dip of the stress arch or else those well to the rise of the line of incidence of the pressure of that arch?

The pressure, be it noted, is not vertical but at a considerable angle. Furthermore, and it is said with diffidence, is not the lifting of the roof after descent and after the destruction of the integrity of the coal—as mentioned by Walter Herd in an earlier paper—due to a movement of the line of incidence of the pressure rather than to a resilient action of the roof due to a sort of wave of force in the rock, such as has been suggested by others? The top bone coal adhering to the roof, as noted by the late Mr. Herd, goes back up with roof when the line of incidence moves forward down the dip, lifting the weight off the coal.

What is underdraw? Is it a general weighting ahead of the pillar or is it a more or less localized weighting due to pressure of the stress arch on the line of incidence in advance of the pillar? It must be a localized stress and not a stress decreasing progressively from the edge of the pillar, or why does the clay bottom move toward the pillar edge? It surely does not move from a zone of lower pressure to one of higher pressure. Perhaps, in all cases, one would more correctly term the incidence a zone rather than a line, for, though it has a line of maximum pressure, it spreads over an area and causes in turn its own tensions, which may be compared to the tilting of the forward end of a short plank resting in mud when a foot is heavily laid on its rear end. Such a "swash plank" is a good example in elucidation of the theory of the middle third.

Speaking of incidence, it must be remembered, as already said, that the pressure of the arch on the coal is not vertical. Let it be supposed that the angle is at 30° to the horizontal, as in Fig. 10, and that at the heading which runs parallel and near the line of incidence is 10 ft. wide. The width of weakness is not merely 10 ft. Some support is afforded if the end of the stress arch strikes the bottom of the coal at the rib of the heading on the stress arch side. Again, some support is provided if the end of the stress arch reaches the top of the coal at the opposing rib of the heading. But these points in plan are not 10 ft. apart but, provided the seam is level, $10 + h \cdot \cot 30°$ where h = the coal thickness. If the coal is 9 ft. thick, the width of weakness is 25.57 feet.

But, as the zone of incidence is in itself a zone of great width, and is no mere line, it is easy to see that with headings near the line of incidence the zone of incidence must make large steps, possibly 100 ft. at a time. Evidently it would be best not to put headings where the zone of incidence is sure to pass later. Surely, it would be preferable to put the roadways within the span of the stress arch—that is, at Springhill, to the rise of the longwall, perhaps in or along the gob of the extracted area—hauling the loaded cars up the longwall face. The pressure being oblique, a heading anywhere else is in danger of collapse due to the pushing out of the coal rib.

Over the resistant part of the pillar, the principle of the middle third does not apply. But the principle holds near the surface for many hundreds of feet beyond the pillar edge. In fact, the line of draw is the line normal to the stress arch at the edge of the resistant part of the pillar, and in the case stated will be 30° to the vertical. Perhaps, then, the greater the span of the arch, the less the angle of draw. I use "draw point" or "draw line" to express the point of maximum tensile stress in an unbroken roof and not merely as a point of subsidence, or even the actual breakage point, which in cases of an earlier breakage may be different, as will be seen later. The depth of the coal being 2000 ft., the distance of the draw point from the line of incidence of the arch will be 2000 × tan 30° = 1155 ft. The draw is a variable quantity dependent on the procession of the line of incidence and on the angle at which the end of the stress arch lies, and these items vary as the stress arch flattens and the soffit rock falls.

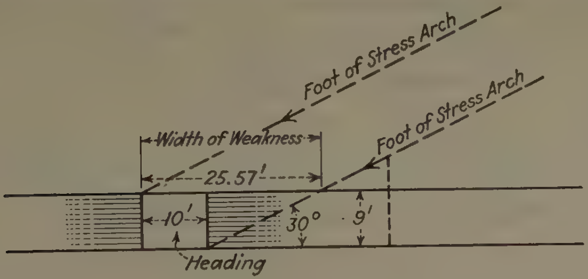

FIG. 10.

Moreover, a premature draw break may make the roof wholly incapable of resisting tension, in which case final draw at the calculated draw angle will not occur. The theoretical draw break is probably often prevented from developing by an earlier draw break, which causes a weak spot which in turn prevents the theoretical draw break from developing.

If the line of incidence is so far back of the edge of the pillar, one should not resist a creep by cribbing at the pillar edge. It may be better to strengthen a heading somewhere within the pillar, nearer the line of incidence. In fact, this is often done. The roof is said to be "throwing her weight." The weight in a well barriered entry may vault the barrier and press on the center pillar, especially where the stresses from two adjacent arches meet. With a narrow excavation, the stress arch will be high in comparison to its width and its limbs will be nearly but, of course, not actually vertical. The line to the draw point, which is at right angle to the stress line at the edge of the pillar, will also be of so easy an inclination, say 30°, that it will be quite long before it reaches the top of the resistant measures near the surface. It will, therefore, develop only a small tension at that point because of the long leverage of the tension. So with such small arches there is rarely any break at the surface. The strain is there but it does not break the rock. Theoretically and in general, draw angle varies inversely as the depth of cover; shallow cover makes large angles, heavy cover makes small angles, but there are many other variant factors, especially whether one is considering a first break, or a second break, a wide span of arch or a narrow one. All the argument, it might be stated, is based on homogeneity of roof, which never actually exists.

T. L. McCall (written discussion).—Mr. Rice has done well in his discussion of this paper to emphasize the difference existing between "district," or "shock" bumps to quote Mr. Rice, and "pressure" bumps. District bumps are doubtless the effect of a sudden readjustment of stress set up by rock rupture in the upper roof. Pressure bumps are presumed to be the resultant of a sudden collapse of pillars in the neighborhood of the active workings.

Mr. Rice suggests that the proximity of the retreating longwall faces to the not fully settled room and pillar workings may be responsible in part for the pressure bumps as described in the paper, and, furthermore, that the long steps between the individual longwall faces may also play a part in making conditions conducive to bumps.

As regards the first of these suggestions, I had formerly entertained hopes that the narrow or ribbonlike extraction was partly responsible for the bumps on the 5700 and 5900-ft. levels, as described by the late Mr. Herd, but the advance of the 6300 and 6500-ft. longwall faces keeping contact with the wastes of these levels formed a waste opening over 900 ft. wide. It is scarcely conceivable that the overlying sandstone bed, at this point somewhere about 30 ft. thick, could possibly be strong enough to span such an opening.

In support of this statement an instance on the west side of the mine may be given, where an area of longwall measuring only 450 by 450 ft. was held to be responsible for breaking the overlying measures and permitting an inbreak of water to take place from the flooded waste of No. 3 seam lying 400 ft. vertically overhead. Furthermore, at this point the sandstone bed, to judge from borehole records, was at least 70 ft. thick.

Regarding the long steps between the longwall faces, I am in complete agreement with Mr. Rice that closely stepped faces, making for all practical purposes one long face, would be infinitely preferable to the present practice, not only as regards the possibility of reducing the bump hazard but also particularly from the angle of the cost of mining.

Unfortunately lack of sufficient development has hitherto prevented this very desirable close alignment but now, with more development available, it is proposed shortly to bring certain faces into close step.

I cannot agree with Mr. Rice in thinking that the immediate roof, by which I assume he means the 14 ft. or so of shale overlying the coal, can of its own weight have any bearing on the recorded convergences of roof and pavement, but the strength of this shale certainly does prevent it breaking and crumbling around the pillars and enables it to transmit the full weight of the superincumbent strata.

Replying to Mr. R. Dawson Hall's discussion, I am of the opinion that although arching may or probably does take place in the upper roof, a very marked distinction must be made between arching and the spanning of an area by a strong sandstone bed in the manner of a beam. This beam action is held to be responsible for the pressure bumps, and the shock bumps also are doubtless the result of overloading and sudden collapse of such a beam in the upper roof.

The most easterly workings in No. 2 mine at Springhill have never experienced any serious bumps, and it is only when the retreating longwall faces enter under the growing sandstone bed that bumps begin to occur.

G. S. Rice (written discussion).—Mr. McCall's reply to the discussion has further elucidated the facts relating to bumps in the Springhill mine, where the mining and physical conditions seem conducive to bumps and to be peculiarly complicated for arriving at a solution of their mechanism and the prediction of a specific occurrence.

We are in entire agreement that the sandstone stratum is responsible for the bumps, and in my observations elsewhere of the bump phenomenon in coal mines I have found that in districts where bumps occur there is always a strong, massive stratum of sandstone or conglomerate above the coal. I use the word "massive" to indicate that the rock does not consist of a series of thin layers. A thin-layered stratum permits successive bending or breaking, even with relatively rigid rocks, of the individual layers over mining excavations without causing bumps. In the majority of coal fields—at least those of Carboniferous age—the greater part of the coal-measure formation is shale or shaly sandstone, which are easily sheared. Hence bumps are practically unknown in most of the great producing coal fields of the world, regardless of depth of workings.

There is always uncertainty in what way and where a conjectural beam of a massive stratum will break by tensional rupture, whether this beam or plate is a cantilever over the solid or is supported on two or three sides. Hence the point where a "shock bump" may strike is uncertain.

Unfortunately I did not have the copy of Mr. McCall's paper when I wrote the discussion. As he states that there is only 14 ft. or so of shale overlying the coal, I agree that it is not probable that this would cause the convergence of roof and floor which Mr. McCall measured. I can only conjecture that there is another factor which we have not discussed; namely, that the convergence was due to the general weighting of the overburden and concentrated by a sandstone stratum, which may or may not have been prefractured or cut by faults at some place far back over the solid coal so that it acts as a separate beam or plate.

As I understood the description of the method of measuring the convergence, Mr. McCall did not determine how much of this was due to the downward movement of the roof and how much to the upward movement of the floor in the levels, as was done in some British studies of convergence.

As concerns the magnitude of the convergence, I suggest, as I did in discussing the matter with the British investigators, that there is a question how far this convergence extends on either side of the passageway or heading where the measurements are made. The shale roof, and perhaps to a lesser extent the floor, will easily deform under such pressure, owing to depth of 3000 ft. or more, as at Springhill, and this is evidenced by the lateral movement of the coal of either rib into the level. In fact, the convergence and flow of the coal appear to follow each other rather closely. To further elucidate my remark, while the convergence of the order of 3 or 4 in. is shown, representing only about 3 or 4 per cent of the thickness of the coal bed before the bump occurred, what degree of convergence would be found, say, 10 or 15 ft. in from the rib? I question whether, if it could be measured, there would be very much convergence found. Nevertheless the method of observing the convergence in the passageways may be valuable in indicating that the hard rock stratum was being put under such tension (either in the lower part if acting as a beam or in the upper part if acting as a cantilever) which was approaching a point of rupture that would cause a bump.

I regret to note that Mr. McCall appears to have adopted the word "underdraw" used by certain British observers, which, as I understand it, indicates lateral movement, caused by pressure, of plastic material under the coal bed. The term is confusing in meaning since the term "draw" has been generally applied to a totally different physical effect, the meaning of which, as generally accepted, is a straight line drawn from a mining excavation to the farthest advanced point of measurable subsidence and not coincident with the irregular line of break (or bend in plastic beds) which extends upward to that point on the surface.

It is hoped that Mr. McCall will continue his convergence studies, as these may provide the means of giving warning of pending disastrous bumps. Further information of value may be obtained by also determining, by leveling, that part of the convergence due to the upward movement of the floor and that part due to downward movement of the roof.

As regards my conjecture that convergence in the body of the coal at a distance from the side of the passageway might not be as much as in the passageway, I suggest it may be possible that the approximate relation might be obtained as follows: (1) Drill a large hole, say, 2½ or 3 in. in diameter 8 or 10 ft. deep into the coal near the roof and another hole near the floor, as parallel to each other and to the bedding as possible; (2) place a strong bar 1½ or 2 in. diameter or strong pipe, in each hole, wedging at the inner end by driving into a short smaller hole, or by packing, and also wedging at the mouth of the hole. Thus in the central part of the hole the bar would have a space around it to prevent it from being bent by intermediate deformation of the coal; the bar to extend a couple of feet or more outside of the mouth of the hole; (3) measure precisely the distance between the bars and the exact inclination of each; (4) repeat these measurements and inclination determinations when measuring the convergence in the passageway. From the data thus obtained calculations may permit arriving at approximate difference, if any, of convergence between roof and floor in the interior of the coal mass as compared with that in the open passageway.

Subsidence from Pillar Extraction at Montour No. 10 Mine Adjacent to the Experimental Mine of the U. S. Bureau of Mines*

By H. C. Howarth,† Pittsburgh, Pa.

(New York Meeting, February, 1933)

This paper gives observations on ground movement and subsidence resulting from pillar drawing in the Lick Run section of Montour No. 10 mine of the Pittsburgh Coal Co. adjacent to solid coal owned by the U. S. Bureau of Mines at its Experimental mine in Snowden Township, Allegheny County, Pennsylvania. The investigation was planned by George S. Rice, chief mining engineer of the Bureau, who arranged the necessary cooperation with Dr. L. E. Young, vice president of the Pittsburgh Coal Co. Engineers of that company placed permanent monuments on the surface to establish the boundaries of the coal owned by the Bureau and to furnish fixed points to which all levels were referred. Engineers of the Experimental mine staff then covered the ground under observation with a checkerwork of monuments made of oak pegs set into the ground at 50-ft. intervals in two directions at right angles to each other. The level of these pegs was determined first from Dec. 16 to 30, 1929; again from May 25 to June 5, 1930, and finally from July 7 to 15, 1931. Observations were then discontinued because work had been stopped temporarily in this part on Montour 10 mine.

General Conditions

Fig. 1 is a plan of the part of the area under observation with which this paper deals. The mine is developed on a room and pillar system known locally as half advancing and half retreating. As Nos. 35 and 36 butt entries were advanced, rooms were driven from No. 36 and drawing of the room pillars followed their completion. When the entries reached the boundary, rooms were started from No. 35 butt beginning at that point, and all remaining pillars were drawn as rapidly as possible. The rib lines of Jan. 9 and June 13, 1930, were surveyed a week to 10 days after the first and second sets of levels were run, and there was some advance in that period. The rib line of July 15, 1931, agrees in time

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† Coal Mine Superintendent, U. S. Bureau of Mines, Pittsburgh Experiment Station.

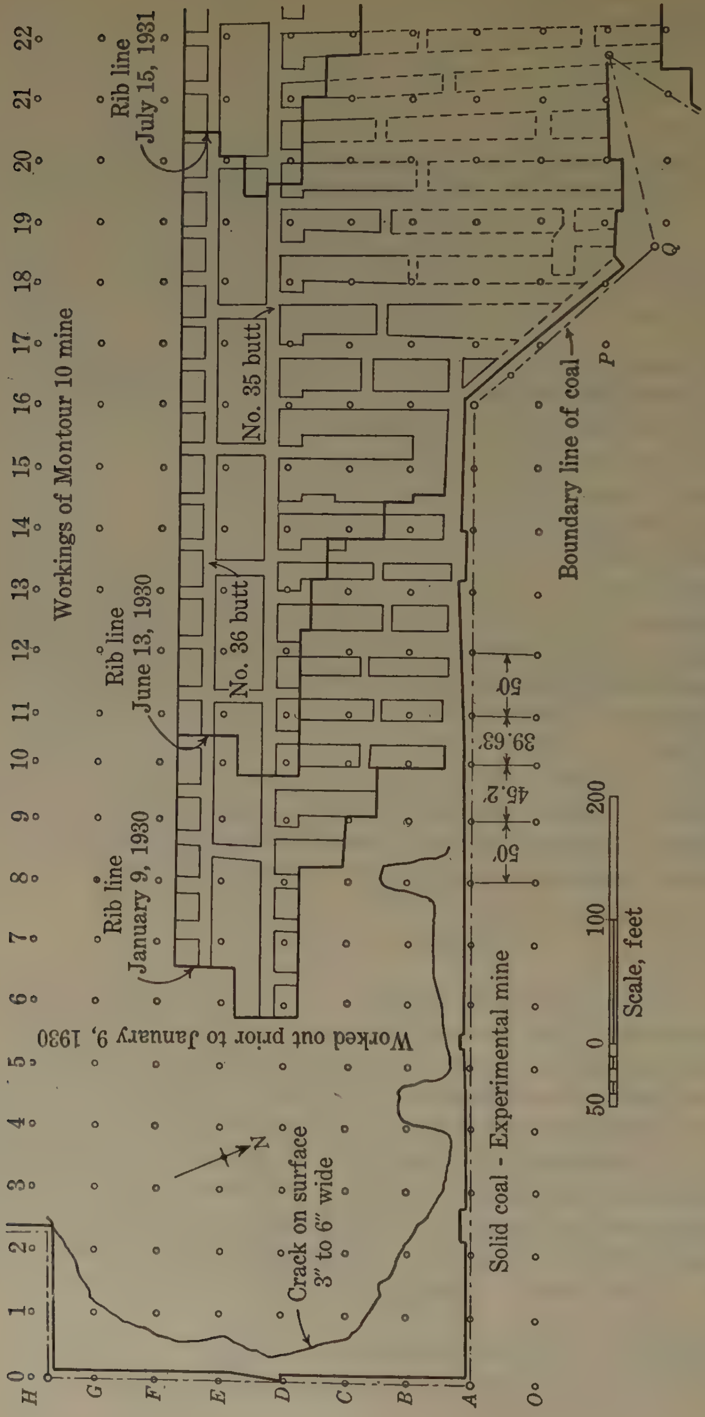

FIG. 1.—GENERAL CONDITIONS OF SUBSIDENCE AREA.

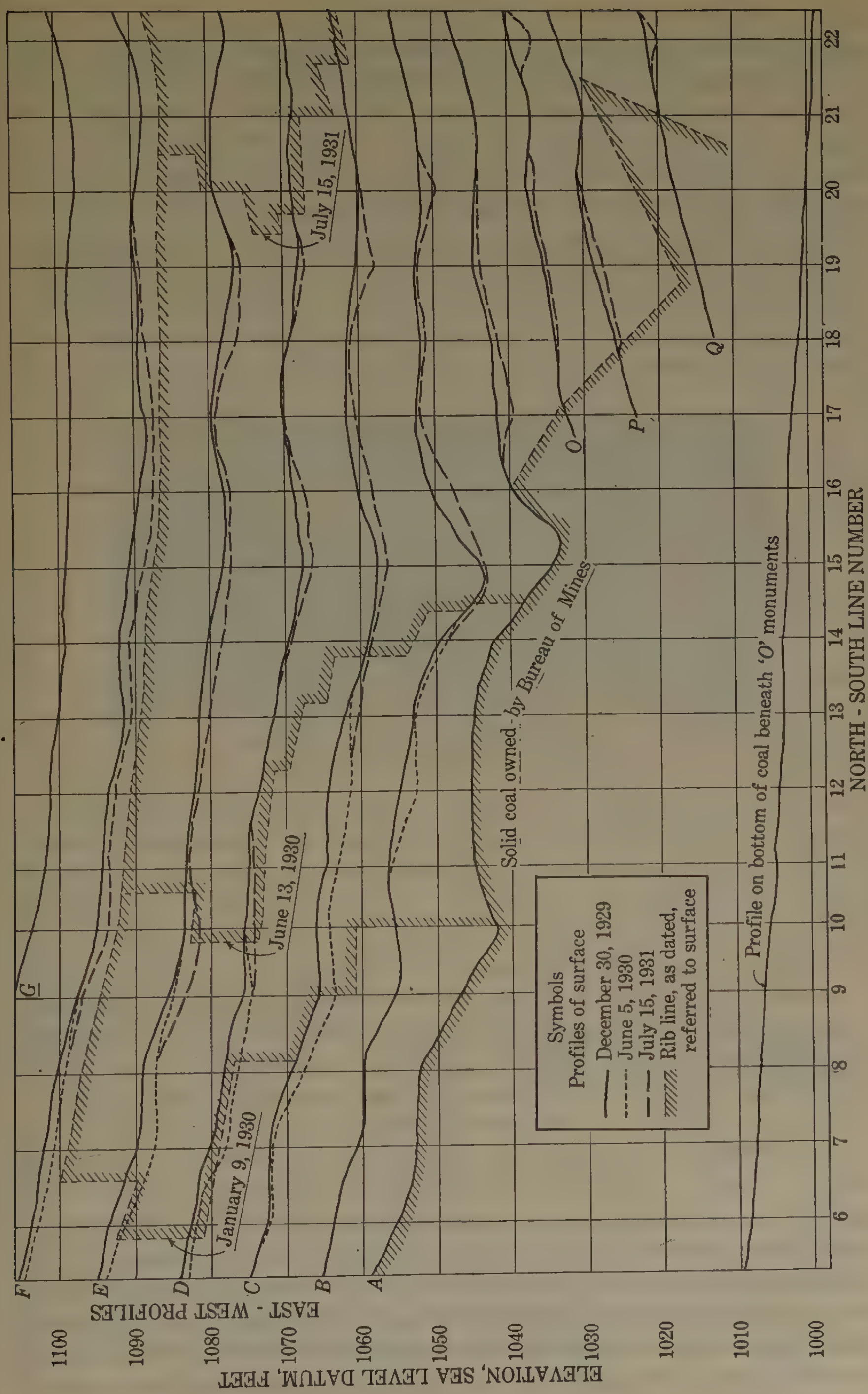

FIG. 2.—PROFILE OF EAST-WEST MONUMENTS SHOWING SUBSIDENCE.

with the final levels. Room work completed at the time of the first survey is shown by solid lines; that completed subsequently is shown dotted.

The average thickness of the Pittsburgh coal bed in this section of Montour 10 mine is about 5 ft. 4 in. The bed dips in a northwesterly direction at a rate of about 1.6 ft. per 100. In the section under observation the coal is overlain immediately by a draw slate above which is a coal roof. The remainder of the overburden is shale, sandy shale, and slate in thin beds with some unimportant thin beds of coal. The subsoil is clay, which in that vicinity is usually found about 4 ft. thick.

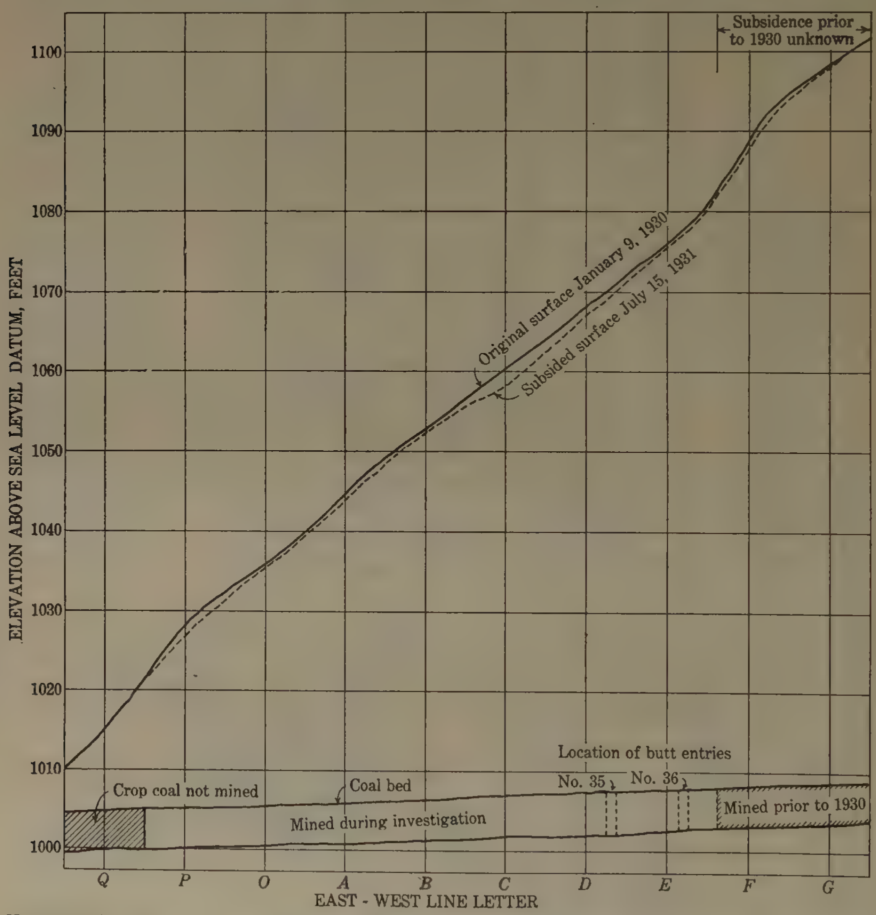

Fig. 3.—Profile of line 19 of north-south monuments showing subsidence (typical north-south profile).

The locations of the pegs for leveling are shown by small circles on Fig. 1. East-west rows were lettered and north-south rows numbered. These pegs were 50 ft. apart except between rows 9, 10 and 11, as shown on the figure. The east-west profile of the bottom of the coal beneath the *O* row of pegs is shown in Fig. 2, together with the profiles of the surface along the east-west rows of pegs above the coal being removed. One typical north-south profile is shown in Fig. 3. The cover over the coal in the area subjected to subsidence ranged from 30 to 90 feet.

SUBSIDENCE

At the start of the investigation there had been subsidence over those portions of the mine from which pillars were already extracted, marked "worked out prior to Jan. 9, 1930" on Fig. 1. This appeared to be fairly uniform over most of the area and reached a maximum of about 3 ft. There was a crack 3 to 6 in. wide roughly parallel to the boundary around the northeast corner of the worked-out area, as shown in Fig. 1. It is probable that the irregularities in outline of this crack along the north boundary are due to the support given by props and waste material in the goaf, and possibly by small unrecovered stumps of coal. Fig. 4 is a view of a break in the surface along this crack between monuments *A*6 and *B*6; at this point it could be examined to a depth of 5 ft. The break was clean and the subsided ground fell away from that over the solid coal, causing no disturbance thereof.

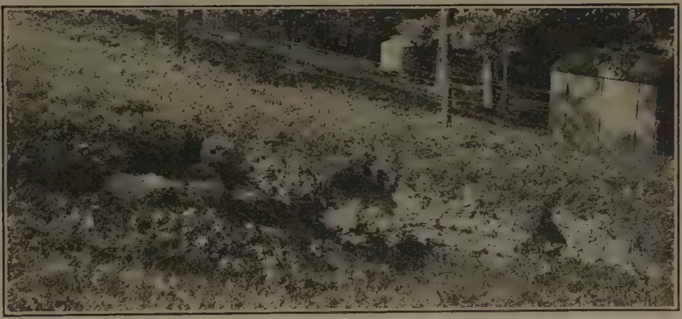

FIG. 4.—BREAK IN SURFACE BETWEEN MONUMENTS *A*6 AND *B*6. COVER ABOUT 40 FT. THICK.

Fig. 2 shows the subsidence that occurred along the east-west lines during the 18-months period covered by the leveling. The solid lines are the profiles at the beginning of the period, the dotted lines immediately below them give the elevations determined by the May-June, 1930, survey, and the dashed lines that continue to the right from the dotted lines show the additional subsidence that occurred to July 15, 1931. The corresponding rib lines are shown with reference to the surface above them and not in their true elevation. The subsidence varied from practically nothing at one or two points to 3 ft. It should be noted that there was no evidence of subsidence along the *A* row of monuments, which were on the boundary line and a very short distance from the actual rib line. Also, subsidence along other rows did not extend as far as the retreating rib line on either of the last two surveys, and, in general, the thinner the cover, the greater the distance from observed subsidence to the pillars. A few pegs indicated upward movement over unmined coal, but investigation showed that these were loose and probably had been raised by frost.

Several local depressions or potholes were formed, mainly at points where the cover was less than 30 ft. thick. They do not appear on Fig. 2, as none of them were on the east-west lines of monuments. At these and other points where the breaks could be examined, it was found that they deviated from the vertical at an angle of about 15°, leaving the surface overhanging the worked-out area.

The coal beneath the *F* row of monuments had been removed about 4½ years before the investigation started. Apparently the ground along this line did not subside completely until additional coal was removed during the present investigation. The thickness of cover along this row was about 100 ft. and the surface overhung the rib line 15 to 20 feet.

The crack parallel to the north boundary shown in Fig. 1 was extended westward as the pillars were drawn, and was 10 to 15 ft. from the *A* row of monuments. There was no disturbance over the solid coal at the angle in the rib line at point *A*16.

Summary

The important finding is that no draw over the retreating ribs or the solid coal along the boundaries was found in this investigation. At the margins of subsided area, in all cases the overburden appeared to arch over the worked-out area. This probably resulted from the thinness of the cover, which did not exceed 100 ft. at any point. It would appear that there is a minimum depth of cover required to produce draw over a retreating rib line, and doubtless this will vary with the nature of the strata. In the present case they were soft and weak.

Local depressions or potholes did not form when the cover was much over 30 ft. thick. Their formation probably depends on the strength of the strata as well as on the thickness of the coal excavated.

Acknowledgments

The writer desires particularly to express his acknowledgments to Mr. H. P. Greenwald, Supervising Engineer of the Experimental Mine Section, U. S. Bureau of Mines, for his advice in the study for and preparation of this paper, which presents only a summary of the comprehensive data obtained in surveys of the monuments.

DISCUSSION

(*George S. Rice presiding*)

R. D. Hall,* New York, N. Y. (written discussion).—The conclusions drawn in this publication are probably fully justified. It is to be regretted that the author did not in his profiles show where "a few pegs indicated upward movement over unmined coal." "Investigation showed," he adds "that these were loose and probably had been raised by frost." One could not expect a rise in the *A* pegs, which are along the fulcrum and where the coal would be crushed down and the clay might slip out and let the coal down, but one might expect the *O* pegs to be lifted. Unfortunately, the elevations of these *O* pegs, which lie a little more than 50 ft. back of the face of the unmined coal, are not indicated. Being over unmined coal, it might be expected that the subsidence of the mined area would cause them to rise. Perhaps they were the pegs on the behavior of which reliance could not be placed.

Work done in the winter will always be under some suspicion as to accuracy because the pegs will lift, and the leveling instrument itself, if not set on pegs driven down to unfrozen ground, well below the surface, will roll unsteadily, as I recall from my own experience. Even in the summer, a level on the surface of the ground has to be adjusted frequently during a long set. Such pegs should be of steel or iron with a short point, all of which should be sunk below frost level, and even such pegs gripped by the frost may be lifted with the buckling surface soil, for the unfrozen soil on the bottom of the peg will not grip the peg as does the frozen soil on the top of the peg. That action is aided by the fact that the end of the peg is pointed and ceases to exercise any grip on the soil as soon as it is pulled loose. The steel peg furnishes less hold than a wood peg, and, if lubricated, is less readily drawn by the frost.

One cannot surmise that much opportunity for "draw" was given except along the line of the *O* pegs on the land of the United States Bureau of Mines, the movements of which are not recorded, and along the center of the entry pillar just north of the *E* pegs. The surface beyond the gob salient at such pegs as *B*16 and *C*15, on June 13, 1930, did not furnish such an opportunity for uplift, because the span is barely 140 ft. and in part much less. Such a span will normally produce little subsidence and, therefore, little uplift over the coal. The marked subsidence noted June 5, eight days earlier, shows that the failure was by shear, a form of failure normal where the cover is light, as in this instance. At *B*12 the cover was about 45 ft. and at *C*12 about 52 ft. At *A*12 it was only 35 ft., a further reason why one should not expect much rise in the *O*12 peg.

Absence of any record of gaping crevices over the area where the stumps were being drawn suggests that there was little, if any, "draw" over that area; perhaps failure to "crevice" is as convincing as the absence of a recorded rise of the surface. It testifies to failure from shear as against failure by bending moment.

My conclusions are: (1) That the winter is the worst time to make such records; (2) that the pegs should, in the winter, be of steel driven down below the frost level; (3) that pegs should be of short point and well lubricated with grease in the frost zone; (4) that levels should be taken with an instrument positioned on pegs protected as described; (5) that bending moment may have caused the 3 to 6-in. crack on the surface in the square made by pegs *C*, *D*, *D*1 and *C*1; (6) that along the line of the unrecorded *O* pegs was where upheaval might have been expected had not the cover been so shallow (at *O*10 not more than 15 ft., for a ravine puts in at this point); (7) that shear rather than bending moment lowered the surface; (8) that open crevicing is an evidence of bending moment rather than shear. A slide may expose a face but it will not make a mouthed crevice.

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* Engineering Editor, *Coal Age*.

G. S. Rice,* Washington, D. C.—The later plugs were set below the frost line. The inferior data are to be supplemented by later work. However, the facts indicate clearly that there was no subsidence beyond the pillars, which is quite important. The necessity of supporting buildings around Pittsburgh suggests the advisability of back filling.

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* Chief Mining Engineer, U. S. Bureau of Mines.

Caving Chambers in Bituminous Mines

By J. W. Paul,* Pittsburgh, Pa. and J. N. Geyer,* Altoona, Pa.

(New York Meeting, February, 1934)

The measures overlying some coal beds are under stress as the result of geological movements after the measures have been deposited. The removal of the coal in the process of mining induces additional stress in the roof measures. Certain types of deposits have sufficient inherent strength to be self-supporting over definite width of excavations and the stress is adjusted without appreciable movement of the material, to the extent that no part of the material will become detached and fall.

The uplifting of the measures in geological time often results in the establishing of cleat planes or planes of weakness in the measures overlying the coal, and these planes generally have a uniform direction. Measures that have not been subject to uplifts are less liable to have noncompensable stresses, although any method of underground mining results in the creation of stress in the roof measures over the excavations and over the coal pillars. The immediate roof measures extending upwards of 10 to 15 ft. above the coal give the greatest concern in maintaining the roof in the advance work in the solid coal. Immediate measures which are inherently weak often may be scaled and supported for indefinite periods by a system of timbering.

As an excavation is made, the pillars adjoining must take on the weight previously supported by the coal removed. The roof over the excavations takes on the function of a beam and if too weak to be self-supporting it will fail either through shearing or bending, and the roof cavity may enlarge until a self-supporting arch is formed.

As support is removed by taking the coal away, there is a tendency for the roof members to move toward the excavation in an effort to compensate the stress; particularly is this true where there is an overlying bed of clay, which may act as a lubricant on which adjoining members may slide.

Unstabilized stresses, plus the force of gravity, plus the inherent strength of the material, govern the tendency of mine roof to fall when not supported by timber.

In general, most mine roof depresses when the coal is removed. This depression may be in the nature of a sag as the result of bending, or the edges of the pillar may break away or the floor may come up or heave.

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* Mining Engineer.

Rigid supports often fail to prevent slight roof movement, whereas if some form of cushion is placed between the cross beam and the roof it will take up the pressure by its distortion and the roof will remain intact. Soft wooden blocks are especially adaptable for the cushions. Where height is involved, metal instead of wooden ties may be used, thus giving additional clearance for rolling stock or animals.

As evidence of the equilibrium of roof stresses may be cited the practice establishing break lines in pillar extraction. Before the break occurs the stresses are carried over the live workings, but upon the roof breaking the roof stresses are equalized or satisfied and roof falls become less frequent in the live workings. In a measure, the roof of a mine may be considered in a state of flotation, seeking an opportunity to break out at the point of least resistance. Should points of least resistance be established in developing a mine, here is where the roof will find its opportunity to give relief to equalize its stresses. These points of relief may be embodied in the plan of the mine. The principle involved is to so plan the dimensions of the excavation that the roof may have a chance to move without breaking. Bending of the roof is less liable to cause a break than a shear. In narrow places trouble with shearing may result in bad falls, whereas if the place is made wider, the shear is replaced by bending so slight that the roof may be held by timber.

A number of instances could be cited where the widening of headings and rooms resulted in overcoming bad roof conditions; also, where the splitting of a heading pillar resulted in the protection of the roof in the parallel heading. In each case the roof material was allowed either to bend, in one case, or expand, in the second case, resulting in better roof conditions on the haulage heading.

It has been observed in some mines operating in the thick Pittsburgh coal in the coking field of Pennsylvania that bad roof conditions in a haulage heading have been remedied by splitting the pillar between the haulage heading and its air course, leaving a rib of coal 4 to 5 ft. thick nearest the haulage heading. The foremen usually speak of this as catching-up the roof.

The observance of these results no doubt suggested the introduction of caving chambers as a means of giving relief to roof stresses in some of the mines operating in relatively low coal in some central Pennsylvania mines.

Some of the details may be of interest at a mine where caving chambers have had a marked effect in reducing the frequency of falls of roof. In this mine the Lower Kittanning coal bed is mined, and until caving chambers were made the systematic order of the day much trouble was caused by the roof falling in the headings, thus adding to delays, expense and a reduced tonnage. The coal averages 40 in. in thickness, resting on a fireclay, some of which is taken up in the haulage headings only, and immediately on the coal is a carbonaceous or bony shale, ranging from 6 to 8 in. in thickness, which is overlain with shale and sandstone varying from 250 to 500 ft. and in the dip section of the mine, 500 to 600 ft. The measures lie along anticlines and synclinal basins and the measures over the coal have well defined cleats or planes of weakness, which add to the roof hazards. Advantage may be taken in some mines to plan the development so that these planes may be crossed either at right angles or at some specified angle, but in planning a retreat fracture line where several pillars are retreating simultaneously, it has been found that roof breaks are more easily obtained when the line of break is parallel to the cleats or planes of weakness in the roof. Under the heavier cover the difficulty with roof falls occurs while turning off cross entries and before the caving chambers can be started, a difficulty that is arrested as soon as the caving chambers are driven.

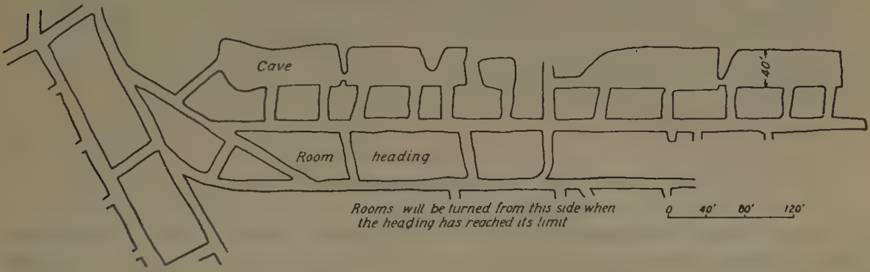
Cave
40'
Room
heading
Rooms will be turned from this side when the heading has reached its limit
0
40'
80'
120'

Fig. 1.—Room headings advancing.

The caving chambers are effective in preventing falls of roof for a distance of 200 to 400 ft., normal to their larger axis, whereas under the heavier cover their effect is lost generally within 100 feet.

Other mines operating in the same coal bed, with similar roof conditions, have tried the caving chamber with much less success, but one probable reason is that the chambers are not planned in advance and their excavation is too long deferred with respect to the advance of the headings.

Fig. 1 shows the plan of a pair of headings advancing with the cave chambers excavated adjacent to the air course. The chamber is kept almost abreast with the air course but better results are obtained when it is kept in advance of the heading.

Fig. 2 shows a plan of two rooms which have been advanced to their limit and the intervening pillar is in course of being removed. These rooms are advanced rapidly and the rectangular pillars are removed during a single shift, thus allowing the roof to cave in the part from which all the coal has been removed.

These rooms are started near the inner end of the panel and on the retreat a slab is taken from the rib, thus leaving a space 100 ft. wide.

One advantage appears apparent; if the roof caves in the 40-ft. wide room, the probabilities are that it will remain intact in the 20-ft. room.

As described by Alexander Jack,[1] the practice at some mines is to locate the caving chambers between the headings. Where as many as four headings are driven parallel the caving chamber is made between headings 2 and 3 and driven 300 ft. in length.

As a means of controlling the roof, it is the general practice to avoid having crosscuts directly opposite in any heading.

The roof in these caving chambers falls in due course and often before they have reached a predetermined distance, and in this event a crosscut is driven from the heading and the chamber construction is resumed after a connection has been made at the face of the caved chamber for the purpose of ventilation.

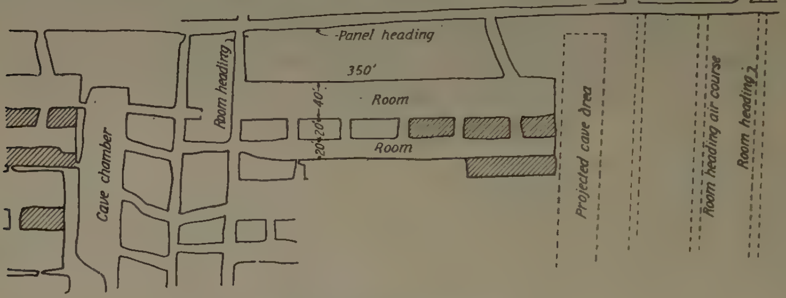

FIG. 2.—ROOM ON RETREAT.

So far as our American literature discloses, a study of the kinetics of mine roof has been too long delayed and a greater study of this feature coupled with a good engineering practice in the layout of mines would lead to less trouble from roof falls and greater economy in mine operation.

The laboratory work done by Professor Bucky[2] at Columbia University is the beginning of a study that may develop into some practical application in the analysis of the roof material of coal mines that may be beneficial in the methods of support of the roof and the prevention of roof falls.

Some of the varying conditions under which mine roof is subjected include dynamic shock from explosives used in blasting coal, change of temperature, which results in expansion or contraction, and seasonal changes in the relative humidity of the mine air. In some mines a heading driven north may have a good roof, while one driven east or west from this may have a very bad roof. If all these factors could be brought under laboratory control, and expressed in terms that would admit of their practical application, the research would be well worth a good-sized fellowship, or at least, some tangible financial support from the industry.

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[1] A. Jack: The Use of Caving Chambers in the Control of Roof. *Proc.* Coal Min. Inst. of America (1931), 102.

[2] P. B. Bucky: Use of Models for the Study of Mining Problems. A. I. M. E. *Tech. Pub.* 425 (1931).

A. F. Brosky's article, "Compressed Air Stages Economic Come-back in Mechanical Mining at Ehrenfeld" (*Coal Age*, December, 1929), gives interesting information on this subject.

DISCUSSION

(*H. F. McCullough presiding*)

G. S. Rice,* Washington, D. C.—The proposal to prepare this paper on caving chambers came through correspondence of Major H. M. Hudspeth, mining engineer of the Safety in Mines Research Board (Great Britain), who wrote to me last October to inquire about further results in the practice of caving chambers which had been described by Alexander Jack, State Mining Inspector at Cresson, Pa.[3] Mr. Jack had stated that "six companies in the Tenth Bituminous district (of Pennsylvania) have resorted to the use of caving chambers with varying degree of success. Several had very satisfactory results and would not try to mine the lower Kittanning seam without their use. Some have had only fair results and others have been disappointed. Conditions vary and identical results from the use of caving chambers should not be expected."

Major Hudspeth, in the letter mentioned, said: "It may interest you to know that a somewhat similar practice is observed at one mine in particular in the North of England and I am particularly anxious to put forward a rational explanation of the phenomenon. It would appear that the driving of one road relieves the stress within a certain radius and that the roofs of the roads subsequently driven within the relieved area are very good. The relief is apparently only afforded if the roof of the road which is driven in advance breaks down. That the caving chambers must be kept in advance of other solid work, if they are to be useful, is in accordance with our experience. It has been suggested that the reason is relief of load which must occur at some distance from where a strong stratum—possibly some distance above the seam—is depressed towards the excavation. The alternative explanation is relief of lateral pressure."

I wrote to Mr. Jack to inquire what additional developments there had been. He replied that there had been no change of any consequence in the use of caving chambers since he presented the paper referred to. He said, however, that a certain company would not try to mine without using caving chambers and if the caved chamber at any time is allowed to lag behind the other parallels, the roof commences to cut in these parallels. He further said that other companies drive caving chambers but not with success. Officials of these companies say that this system slows up the development. He concedes that point but thinks that the advantage gained is worth while.

M. D. Cooper,† Pittsburgh, Pa.—Mr. Jack emphasized the value of caving chambers between the second and third entries in a four-entry system, and that practice is being followed.

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* Chief Mining Engineer, U. S. Bureau of Mines.

[3] Reference of footnote 1. The maps Mr. Jack showed at the presentation and the discussion that followed were not included.

† Division General Superintendent, Hillman Coal & Coke Co.

P. B. Bucky,* New York, N. Y.—Mr. Jack inquired about the possibility of testing this method by means of models. Provided with a good geological section and sample of the materials it should be possible to determine the mechanism, which is evidently a function of the shape, strength and dimensions of the materials above the opening. There are two possibilities: (1) a solid bottom and (2) a yielding bottom with the floor coming up. The strength and thickness of the strata immediately above the coal are of interest. Small-scale models should be tried to determine the mode of breaking.

R. D. Hall,† New York, N. Y.—Caving chambers had their origin in Nova Scotia or Great Britain. The theory that collapses of roof that demand the use of caving chambers are the result of residual stresses, themselves caused in turn by mountain-forming movements of the earth's surface, has rarely been postulated in America. It is held, perhaps rightly, in parts of Great Britain. The expansion resulting from chemical changes in the roof material seems to be a more probable and a more generally held explanation.

When an iron carbonate is changed to a sulfate by sulfurous waters, it expands 2.65 times; a calcium carbonate expands to 1.33 times its former volume. Similarly, if diaspore becomes an aluminum sulfate, it has 11.57 times the volume of the original. Gibbsite expands 6.17 times and bauxite 11.54 times. Of course, the original minerals, when present, are spread through the rock and may not be entirely sulfatized, so the expansion of the rock as a whole is less than these big figures would suggest.

When the rock expands from recrystallization, it presses down on the coal and tends also to lift and bend the overlying measures upward. With an arched opening, the rock may satisfy its need for expansion by expanding laterally into the opening, so that the lateral expansion instead of being equal to the cube root of the volumetric expansion, taking place in one direction only, may be equal to the cubical expansion, and so may be considerable. Usually, movement can find leeway in only one direction, and all movement is thus directed. Expansion does not penetrate far or fast from the exposed surface, and falls may be delayed for months.

At Gallup 12 by 12-in. timbers installed in roadways would be broken in a few days and, thereafter, the expansion was so small that lighter timbers would serve to hold the roadways open. Investigation is needed to decide between Mr. Rice's residual-stress and the expansion theory. The abounding presence of crystals in the roof at Gallup suggests an entirely different condition from that in eastern mines. The crystals probably expand considerably on becoming moist. The Gallup mine would be bone dry except for the moisture emitted from the metabolism of the coal after exposure. It has so much moisture, and what it has is so loosely held, that coal freshly mined crackles as the result of its release. This emitted moisture gives an otherwise dry mine an oppressively moist atmosphere. The coal is subbituminous and is probably derived from deciduous rather than coniferous vegetation, for deciduous limbs are greatly in evidence, and these are preservable only if protected from air, as by a coating of shellac.

G. S. Rice.—Mr. R. Dawson Hall attributes the falls to chemical changes or recrystallization of roof material, but it would appear to me that this could not be the primary reason in the central Pennsylvania mines where falls are reported to occur so immediately. If it had this effect in the caving chambers why would it not in the parallels? Further, the caving chambers, as indicated by Mr. Jack and Mr. Paul, have to be driven in advance of the parallels to be effective. Therefore, I cannot see any other explanation than that in the particular mines in places where falls quickly occur there is a local lateral pressure due either to geological thrusts going on slowly in the particular area or residual stresses from previous folding.

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* School of Mines, Columbia University.

† Engineering Editor, *Coal Age.*

A most striking instance of this lateral pressure I saw in a mine at Gallup, N. Mex., 30 years ago. The mine was a slope mine in the trough of a dipping synclinal fold. My recollection is that the main slope, which was parallel with the fold or overthrust fault, was only ½ mile or so from the edge of the field where the coal measures were upturned nearly vertically. Upturned strata and coal bed cross the A.T. & S.F. Ry. cut a few miles east of the Gallup station. In this mine not only the roof but the coal also was under lateral pressure at right angles to the dip. In the advance headings when the face was struck with a maul small pieces of coal would spall off violently, so that at times one had to get back until the throwing off had subsided. The miners liked this condition because they got their coal easily. In roadways going to the dip—that is, parallel to the fold—it was impossible to hold the roof up by ordinary timbering. The roof would break sometimes 5 or 6 ft. or more above the coal, the exposed roof shales would set over the coal ½ in. or more and thereafter the place would be quiet. The lateral movement was further demonstrated by the breaking of cross timbers or struts from rib to rib by the side pressure. After this situation developed, in mining deeper from the outcrop, no attempt was made to turn rooms in any block or panel until headings on the sides had been driven up or down to the next pair of levels, so that in effect these slip or raise headings were caving chambers.

In rooms subsequently driven to the raise, parallel to these headings, there was no special trouble in maintaining the roof beyond the usual necessity of systematic timbering for maintaining a shale roof.

As I understand Mr. Hall's comments, he seems to consider that the conditions of the roof in Gallup mines which led to immediate falls in dip or raise chambers was general throughout the field. I do not think it was. The roof, although shaly, was reasonably good in most places. Probably the particular mine to which I refer was worked out or abandoned before Mr. Hall visited Gallup. When I was acquainted with the Gallup district, 25 to 35 years ago, through frequent visits to the mines, it was not necessary to use 12 by 12 timbers, which he speaks of. Their use may have been required later as the mines got deeper but the cross timbers that I spoke of as being broken by lateral pressure in advance headings (in the most easterly of the mines) were of ordinary size, about 8-in. diameter round timber.

L. E. Young,* Pittsburgh, Pa.—I would urge everyone who knows of attempts to use caving chambers, either in bituminous coal or anthracite, to inform the chairman of the bituminous mining methods committee, so that adequate records may be compiled.

J. J. Rutledge,† Baltimore, Md.—A professor of the Royal School of Mines, while on a visit to this country a year or two ago, made some investigations on the effect of cleat on mining operations and visited some of the mines near Cresson, Va., in which the caving plan was being followed. This gentleman gave an illustrated lecture before the Holmes Safety Association at Cresson.

In the mines near Springfield, Ill., where there were numerous "horsebacks" or clay seams cutting vertically through the coal seam, it was possible to hold the roof in panel entries driven 20 ft. wide, but not in such entries when driven 9 or 10 ft. wide.

A recent article[4] described a plan somewhat similar to that described by Mr. Paul, in the Parkgate seam, where conveyors and panel longwall had been replaced by post and stall. In the Big Vein mines in Maryland, in recovering pillars, the pillar is never split except under unusual circumstances, such as when the cover is very light, but the plan usually followed, and successfully, is to slab up the side of the pillars, using sufficient timber, and in this way to recover the coal. If the thick coal in central or southern Illinois is ever remined, some such method doubtless will be followed.

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* Vice President, Pittsburgh Coal Co.

† Chief Mine Engineer, Bureau of Mines, State of Maryland.

[4] *Iron and Coal Tr. Rev.* (Jan. 12, 1934).

MEMBER.—Has this method been tried in the anthracite region near Scranton?

A. E. YETTER,* Scranton, Pa.—To my knowledge, this practice has not been followed in the anthracite region near Scranton. In beds having a fireclay roof, the moisture in the air is soon absorbed by the roof rock, causing the roof to swell and crumble and finally to fall into the open chamber. Unless a fairly good percentage of the original coal (35 per cent or more in thick beds) is allowed to remain in uniformly distributed pillars, squeezes are likely to occur, either before or during pillar extraction. This, of course, applies to any bed of coal whether it has a fireclay roof or not. Wherever possible, it is more economical to arrange the first mining so that the pillars can be recovered soon thereafter, thereby eliminating extra expense of handling fallen fireclay roof rock, retimbering, etc.

L. E. YOUNG.—Difficulties have been encountered with old rooms in which draw slate has been packed. These are now a serious obstacle to mining, which raises a question as to what will happen under the caving system when an attempt is made to recover pillars on the retreat, in the caved areas.

E. MCAULIFFE,† Omaha, Neb.—The Gallup coal is a hard unfissured subbituminous coal, which has no opportunity to absorb stresses. Hence it is under lateral pressure. The situation in Wyoming is somewhat different; there they have a laminated sandstone roof, which contains insufficient lime to create chemical activity, but which slacks from moisture. Temperature changes also induce roof falls, so that it is necessary to drive workings rapidly if the roof is to be properly maintained. If a room can be completed in three weeks, little trouble is to be expected. In Wyoming the earlier return air courses were hard to keep open but when the air current was reversed, the flow of dry air through the moist return air courses set the sandstone roof like concrete.

R. L. AUCHMUTY,‡ Pittsburgh, Pa. (written discussion).—Caving chambers are commonly used in Central Pennsylvania, particularly in the B, or Lower Kittanning, seam, generally under heavy cover, 500 ft. and over. The seam is generally pitching with grades sometimes up to 18 per cent. Many mines in that section are developed from slopes driven down the pitch of the seam from the outcrop. The B seam is about 40 in. thick, on top of which is about 4 in. of bone coal overlain by about 5 in. of draw slate. The main roof is a strong slate or sandy shale, usually over 15 ft. thick, underlying a thin seam of rooster coal overlain in turn with sandstone. The bottom is a slate parting a couple of inches thick under which is a few inches of dirty coal underlain by a thick bed of fireclay. The seam frequently is cut through with clay veins and the bottom is very uneven and very rolling. At times the rooster coal over the seam comes down very close to the top of the main seam.

Bad top conditions are generally encountered in the vicinity of clay veins and also where the rooster coal with its sandstone top comes down close to the main coal seam.

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* With James H. Pierce & Co.

† President, Union Pacific Coal Co.

‡ Eavenson, Alford & Hicks.

However, when workings advance to the point where the cover is 700 or 800 ft. bad falls are apt to occur under what appears to be very favorable top conditions.

Under this thicker cover heavy falls of slate are not infrequent in advancing headings before rooms have been turned in the midst of a large area of virgin coal. Apparently width of place or direction has little effect on the occurrence of these falls. Generally these caves come without warning shortly after the face has advanced. The top apparently will be sound and the following day will have fallen the full width of the place, regardless of how heavily the place may be timbered. Frequently it happens that when one of these heavy falls occurs on one heading there will be no trouble in the parallel entry. For this reason an extra entry is driven wide, as illustrated in the paper, for a caving chamber in the hope that falls will come in this entry and not on the haulage entry. This is partly successful; if a fall comes in either the air course or caving chamber, one or the other can be used for air and thus avoid cleaning up the fall. The caving chamber driven on the low side is also frequently useful for drainage purposes.

The fact that these falls come when entries are being advanced in solid coal does not necessarily mean that there will be bad top conditions when rooms are driven and pillars are mined. When pillars are extracted in this same seam it is not uncommon to see rather large areas from which all the coal has been removed standing for some time supported only by a few props. On pillar work there is more apt to be trouble with bottom heaving than with bad top, although with a reasonable rate of extraction there is little difficulty from this source.

It would seem that the theory of beam action of the roof in solid places, driven under sound top, even under heavy cover, would have little to do with the unusual action of top under these conditions. Possibly internal stresses in the immediate roof strata due to folding is the most important influence.

I do not think that this is a problem that lends itself to solution in a laboratory with test pieces of rock from the roof strata. More detailed study of conditions underground would probably lead to more positive results.

J. W. Paul (written discussion).—The recrystallization of the material in the roof members, as proposed by R. Dawson Hall, does not appear to be a dominant factor in inducing the shearing, bending or breaking of the immediate roof. If his theory were true, there would be a manifestation in the roof of mines developed in measures lying horizontally and not under the influence of residual stresses as the result of previous folding.

The essence of the problem is that stresses are either residual or they are set up as the result of the excavations made in mining the coal, and it is these stresses that are given relief by the use of caving chambers, the increasing of the width of headings or rooms, or the driving of additional parallels, which give opportunity for the roof either to bend without breaking or to shear and fall in one of the excavations. The problem is not one to prevent inherent stresses, or stresses set up by chemical action, but one to adopt a practice in mining that will minimize the occurrence of falls of roof that otherwise add to the hazards and cost of operation. The proper use of caving chambers, or other excavations that give similar relief, may be of great advantage in many mines, but their planning needs a study of local conditions and engineering guidance.

The Dedusting of Coal

By Henry F. Hebley,* Chicago, Ill.

(Hazleton Meeting, October, 1932)

In recent years, especially in the last decade, great interest has been shown and many advances have been made in the preparation and cleaning of coal. In the major coal-producing countries, the percentage of tonnage cleaned has been increasing ever since 1924.

As separating media, both liquids and gases are used, these generally being water and air respectively. Some systems, instead of obtaining separation by the velocity of the medium in which the coal is cleaned, depend on the density of water mixed with sand, or coal dust and shale, or with chemicals such as calcium chloride.

In studying the results of coal-cleaning equipment, it will be observed that the efficiency of the cleaning apparatus lowers as the size of the coal decreases.

This inability to obtain any marked improvement in the fines has been partly responsible for the installation of dedusting equipment, to remove the fine sizes from the raw coal prior to cleaning.

When considering the possibility of installing dedusting equipment, many factors must be taken into account. Very often, but not always, both the ash and sulfur content of the coal increase as the coal size decreases. When bituminous coal, containing soft powdery fusain, is subjected to a wet washing process, the porous fines of the fusain remain in the small coal or in the wash water and delay the dewatering of the washed coal and increase the difficulty of clarification in settling tanks or thickeners. When a dedusted coal is cleaned by a wet process, the sludge recovered from the settling tank often runs so high in ash that the sludge can be discarded without further treatment and still the loss is small. Where treatment of the sludge is desired, however, froth flotation may be used to advantage if the size of the feed is below $\frac{1}{10}$ in. Dust from the deduster can be treated in a similar manner. Fusain, which is usually found in the fine sizes, has often a much larger ash content than other coal.

Illustrative of this condition is a table of ash contents for the various petrographic constituents of Illinois coal (No. 5 bed) as given by Mitchell[1] (Table 1).

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* Allen & Garcia Co.

[1] D. R. Mitchell: Coal Preparation Problems in the Illinois Field. *Trans.* A.I.M.E. (1931) **94**, 301, Coal Division.

TABLE 1.—*Ash Contents of Constituents, Illinois No. 5 Coal (Mitchell)*

Name	Percentage of Ash
Clarain	4.8
Durain	11.9
Fusain	34.4
Pyrite	59.2
Clay	69.5
Shale	74.2

When clay or soft shale is present in the fines, these impurities tend to disintegrate when brought in contact with water, lengthening the drainage period and usually increasing the ash content of the clean coal.

With clay in the washery water, there is much difficulty in the clarification of the water and in the dewatering of the sludge after recovery from the settling equipment. This makes large settling capacity necessary. With dedusted coal, however, the size of the settling equipment can be reduced.

When studying a coal for a byproduct market, further influencing factors must be considered. As is well known, the mixing of wet fines with cleaned coal destined for coke ovens raises the moisture content. The resulting increase in coking time and in maintenance due to deterioration of the brickwork is appreciable. If fusain is present its percentage in the mix should be controlled or a good hard coke will not be obtained. Uncontrolled fusain often causes coke breeze in the manufacture of coke, as this material is mineral charcoal and is practically unchanged during carbonization.

TABLE 2.—*Prevalence of Dedusting*

Name of System	Type	Dedusting Used
Wet		
Simon-Carves	Baum jig	Yes
Norton-Harty	Baum jig	Yes
Rheolaveur	Trough	Sometimes
Chance	Density liquid	Yes
Lessing	Density liquid	Yes
Dry		
American Coal Cleaning Co	Air tables	Yes
Birtley	Air tables	Yes
Demag	Air tables	Yes
Bemag-Meguin	Air tables	Yes
Air-sand	Trough density	Yes
Peale-Davis	Air tables	No
Raw static	Air tables	No
Kirkup	Trough	Yes

In the cleaning of coal with air as a medium (*i.e.*, dry cleaning) the dust is lifted to the top of the bed and most of it is lifted into the air and redeposited with the clean coal. Consequently, all air cleaning systems with the exception of the Peale-Davis, and the George Raw process generally remove the dust prior to cleaning. This is also true with many of the wet coal-cleaning systems, as is shown in Table 2.

Another criticism of dirty fine coal has come to notice recently with the adoption of small forced-draft automatic underfeed stokers which have been installed in numbers under the low-pressure heating boilers of American hotels and apartment buildings. In the middle-western coal field, the ash fusion point is generally low and when these coals are burned on small underfeed stokers the stoker tends to lift the ash to the top of the fuel bed. In this hot zone the ash either slags or is lifted by the air velocity into the first pass of the water or fire tube boilers. Here the small particles of molten ash freeze to the tubes and gradually increase in size, reducing the draft and lowering the efficiency of the boiler.

To clean such a boiler effectively, it is necessary to take it off the line, which increases the "outage" of the unit. By experiment it has been found that coals with the fine dust removed do not lift off the grate so easily and the trouble from "clinkering" is reduced. Also, the fuel bed using this prepared coal is more open, and as most mid-west coals are "free burning," the air for combustion is more uniformly distributed over the grate area.

If after investigation of a specific coal it is found desirable to dedust it, a second problem arises in the disposal of the dust after it has been removed. Until recently the loading, transportation and unloading of fine dust has been a difficult problem, owing to the lack of properly designed equipment. Now, however, in Germany, America and England, special tank cars have been constructed, which eliminate these difficulties.

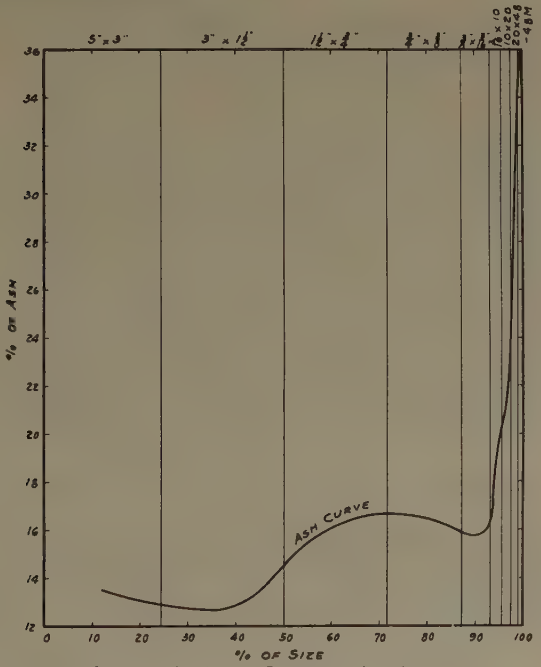
When considering a market for coal dust, its possible use as "pulverized fuel" naturally presents itself. Assuming that it has a fair calorific value, it should be in great demand for this purpose, although further grinding may be necessary. Since the trial run of the H.M.S. *Scythia* of the Cunard Line, colloidal fuel, which was quite an intriguing subject during the later years of the war, has again come into prominence, and if further experiments prove successful, the dust would find a new market, for it could easily be prepared and pumped into tank cars for shipment to any plants, both land and marine, that desired it.

Briquets made of the dust, with a suitable binder, form another outlet, especially for the domestic trade.

It has also been suggested, and subsequent experience will prove whether it is feasible, that the coal dust may be used in the Heller process for the manufacture of blue water gas. This system is well described by Thau.[2]

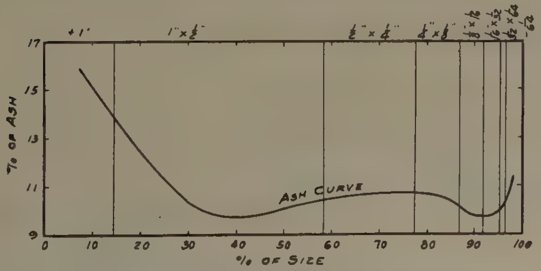
Before introducing dedusting equipment, a screen analysis of the coal with the corresponding ash contents of each size should be made and studied, as the ultimate treatment and disposal of the dust depends in great measure on this study.

Fig. 1.



Illinois LaSalle County No 2 Coal

Fig. 2.



Thick Freeport Seam • Pennsylvania

Figs. 1 and 2.—Variations of ash content with coal size.

The curves in Figs. 1 and 3, representing coals from different seams and districts in the State of Illinois, show a rapid increase in ash content as the sizes decrease. This is true of many seams of Illinois and Indiana, and indicates that the coal could be dedusted to advantage. The dust on certain coals may be so dirty that it would not be clean enough to mix with the coal from the cleaning plant. In that case it may be possible to use the dust at the boiler plant of the mine, if the mine generates its own power.

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[2] A. Thau: *Proc.* Third International Bituminous Coal Conference (1931).

In Fig. 2, ash percentage and size decrease concurrently, so there is no reason why the dry dust should not be mixed with the cleaned coal (be it either wet or dry) for boiler use. If the coal is to be coked, the coking properties of the dust must be investigated, to ascertain whether the dust will have an adverse effect on the coking action of the coal. Mixing the clean dry dust with wet cleaned small coal is distinctly advantageous because it reduces the moisture content of the fuel. Further consideration of the curves previously mentioned, together with a close inspection

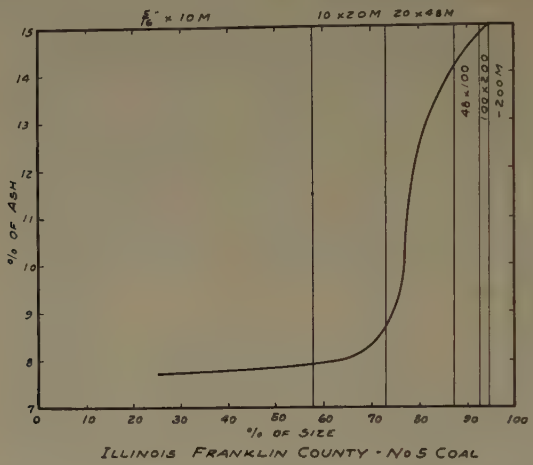

FIG. 3.—VARIATION OF ASH CONTENT WITH COAL SIZE.

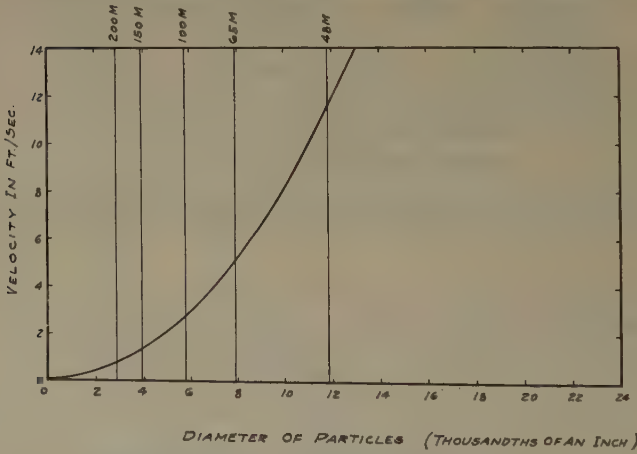

FIG. 4.—ULTIMATE FALLING VELOCITIES OF PARTICLES IN AIR.

of the actual coal on which the curves are based, bring interesting facts to one's notice. The impurities of some coals are hard and do not easily break into small pieces, and the dust from such a fuel consists of fine soft coal low in ash. On the other hand, coals that are hard and contain soft clay and shale will yield a dust with a high percentage of ash.

Thus dedusting is a treatment to be kept in mind when considering cleaning and preparation of coal.

Dedusting Equipment

Two major problems face the designer when coal has to be dedusted. First, there is the difficulty of removing a predetermined size of dust, without carrying over with it any more larger material than is absolutely necessary. Similarly, one should strive to remove all the dust possible out of the oversize. To word it differently, the "cut" should be made as sharp as possible. Many factors influence the success of this aim. The fines of the coal may contain a high percentage of free moisture; the coal may carry as impurity soft sticky clay which clings tenaciously to the larger material or tends to "ball," holding several particles together, and the particles of shale may be laminated in shape, affording a large surface area with small mass.

Dust is usually removed from coal either by screening or aspiration, and in some equipment both methods are used.

The second problem is that once the dust has been extracted from the coal means must be provided for its collection. The agencies by which this collection is accomplished are generally: settling chambers, settling chambers with sprays of steam or water, cyclone dust collectors, bag houses, and cloth filters either of the frame or tube type.

Various designs of dedusting apparatus are available, which will be described in the following pages.

As shown in Fig. 5, taken from Minikin,[3] the raw feed passes over a 10-mm. screen, the undersize being collected in a storage bin. A rotary feeder drops the minus 10-mm. particles at right angles to a blast of air so controlled by velocity and a vertical baffle that coarse dust of the desired size is deposited in one hopper while the 3-mm. dust is driven over the baffle into the fine dust hopper.

In the Buhler dust extractor[4] (Fig. 6) the coal is passed over a series of cascades and the air is driven together with the dust from the top of the coal stream, through the louvres into the coarse-dust chamber, thence, after the settlement of the coarse particles, the air and fine dust pass to the suction fan and the cyclone collector.

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[3] R. C. R. Minikin: Modern Coal Washing Practice.

[4] Reference of footnote 3.

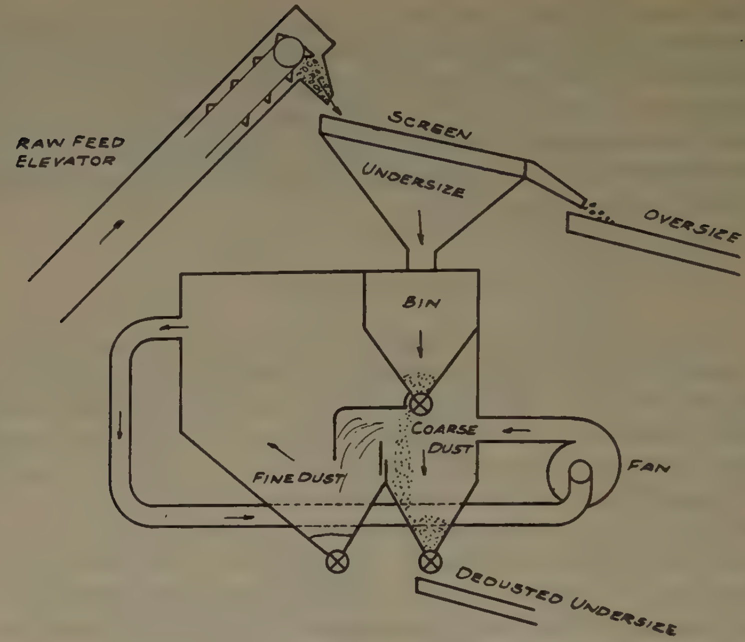

FIG. 5.—MEGUIN DUST EXTRACTOR.

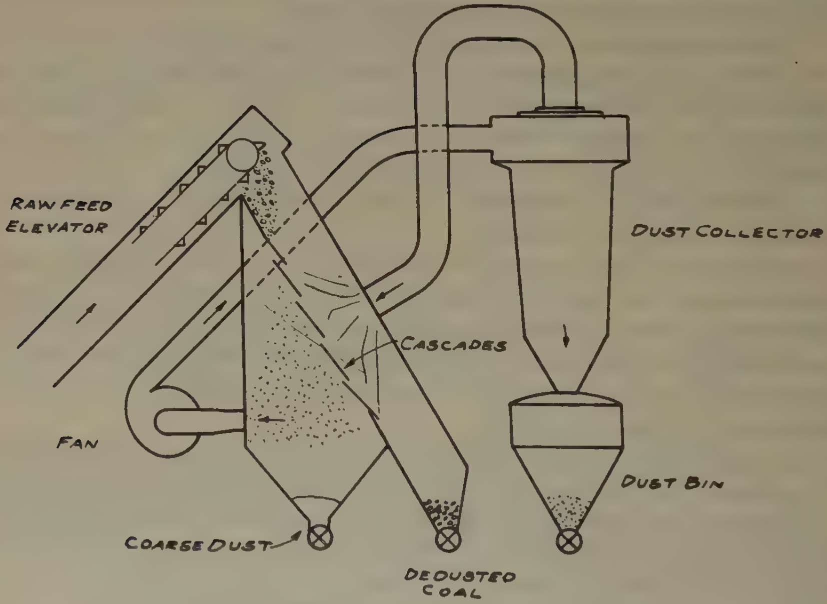

FIG. 6.—SIMON BUHLER DUST EXTRACTOR.

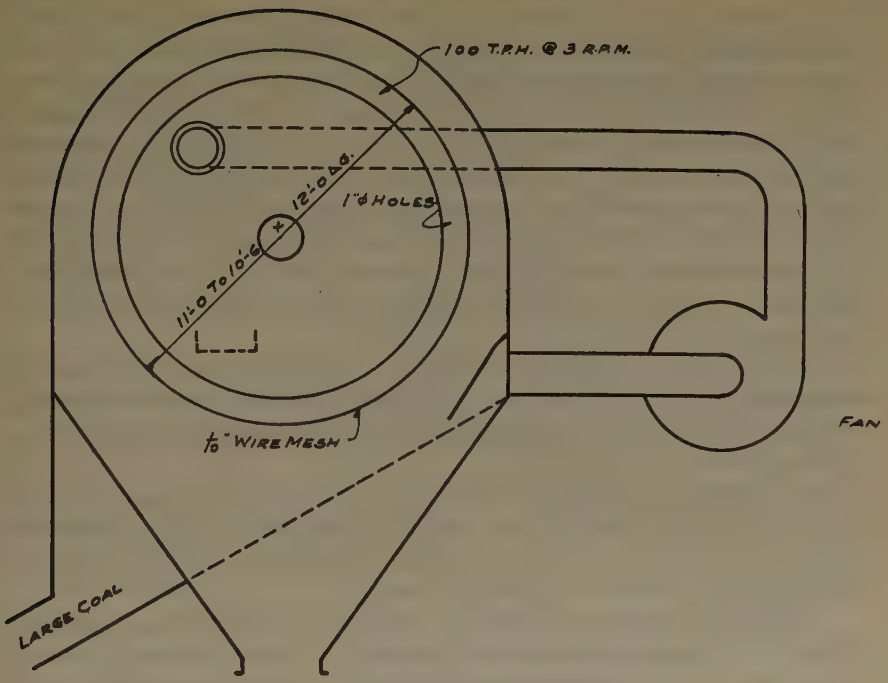

FIG. 7.—REVOLVING-SCREEN METHOD.

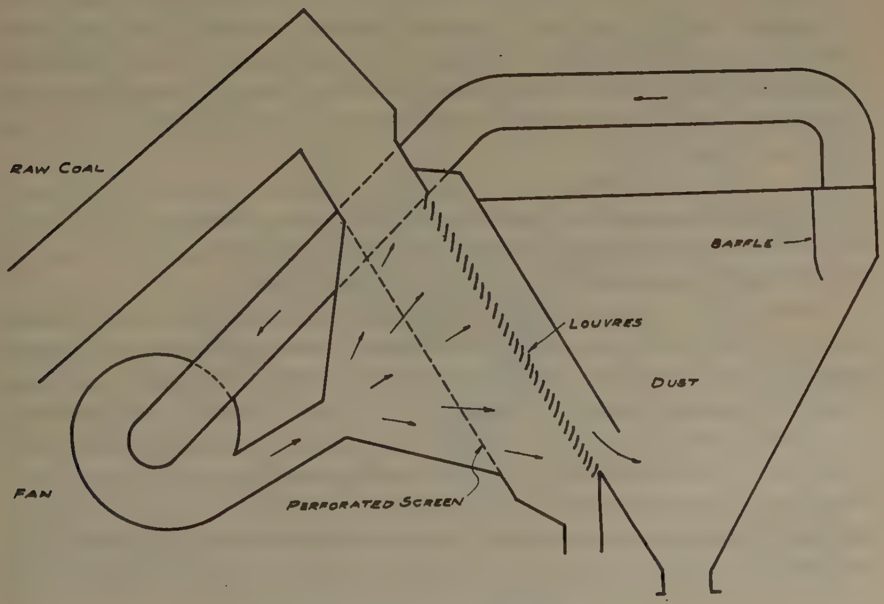

FIG. 8.—MINIKIN DUST PLANT.

In another system, using a concentric revolving screen[5] (Fig. 7), with 1-in. holes on the inner screen and 1/10-in. wire mesh screen on the outer circumference, the coal is divided into plus 1 in., 1 to 1/10 in., and dust, the last of these settling into the dust hopper. To facilitate the separation, air is circulated through the system. The Minikin dust extractor[6] (Fig. 8) is the reverse of the Buhler type. It has been noted that when small coal passes over a sloping perforated plate the fines rise to the surface. Therefore, in this design the air blast is placed below the coal bed and blows through it, lifting the dust through louvres into the dust hopper. The feed to the dedusting plant is 3/8 to 0 in. and minus 2-mm. to 0 dust is extracted.

In all the foregoing designs the air is recirculated and, if hygrometric conditions are favorable, the relative humidity of the circulating air may build up to such a point that trouble will be caused by water being deposited with the products.

Fig. 9 shows an American deduster known as the Blaw-Knox, in which two adjustable air nozzles direct streams of air against falling curtains of coal. The dust, having been driven out of the coal, is drawn off by two fan suctions and delivered to dust collectors.

The Hunter dedusting apparatus (Fig. 10) is similar to the Buhler but has also rollers or agitators on the slope and adjustable louvres to control the velocity (and consequently the size) of the various dust products desired.

The Lessing system (Fig. 11), a type that has been quite successful in England, allows the coal to meet an upward current of air and by a control of its velocity dust of the desired size may be removed. With this equipment the air can be controlled accurately and by arranging more than one of these units in series a greater degree of control over the various sizes can be provided.

In an attempt to overcome various difficulties, such as the "blinding" of small mesh screens when handling coal of a wet or sticky nature, and to effectually scrub any small particles adhering to the larger sizes of coal, the dedusting system shown in (Fig. 12) was devised.

With this equipment, which provides both screening and aspiration, the raw coal is fed through an airlock on to a double-deck vibratory screen of circular shape. The upper screen, having the coarser openings, has the opening dimensions fixed by the type of coal to be dedusted. The lower screen has a finer mesh proportioned according to the coal. The coarse material falls in a circular discharge and is cut by the air rising from the dedusting tube. The dust-laden air passes through a set of conical louvres and passes down the central tube to the dust-collection apparatus. To prevent the "blinding" of the screens, two rotating arms

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[5] Reference of footnote 3.

[6] Reference of footnote 3.

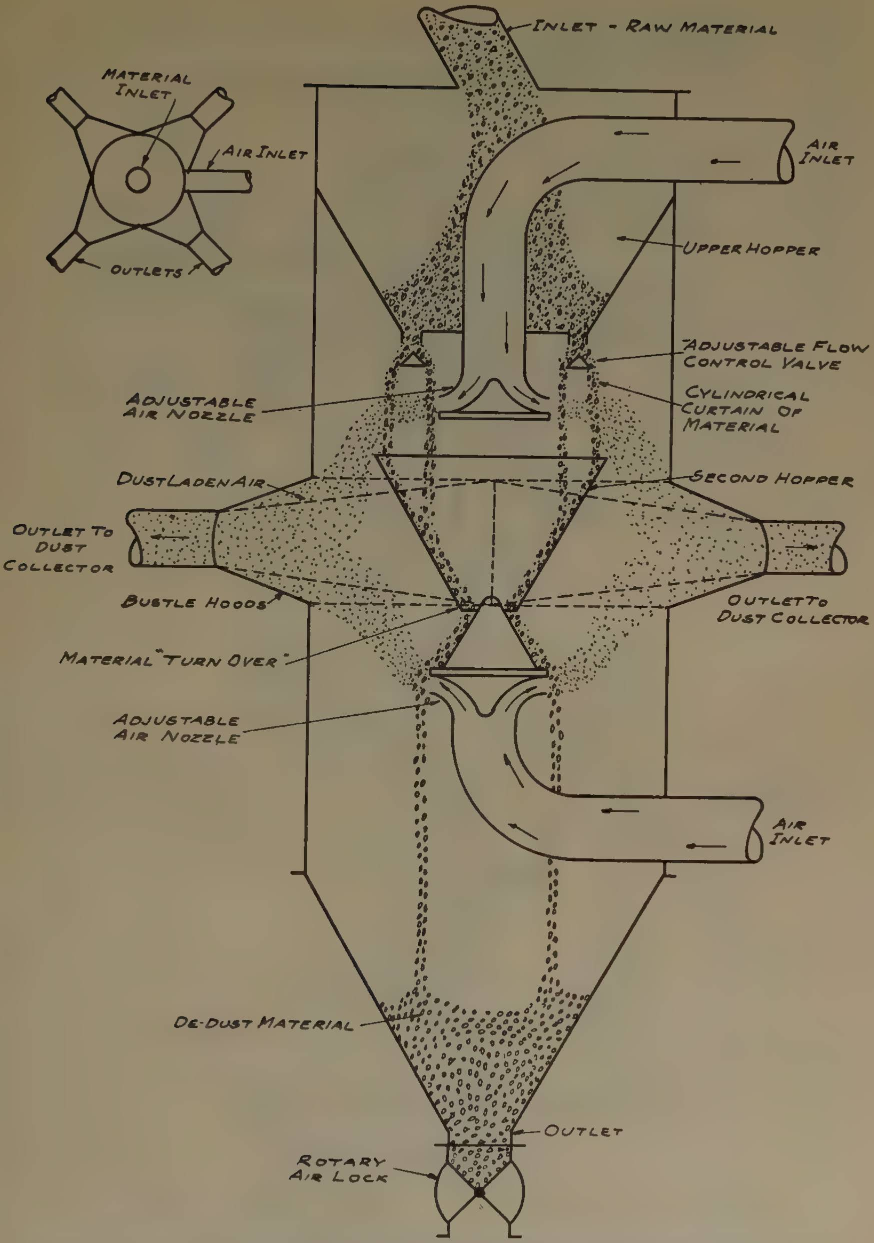

FIG. 9.—SECTIONAL ELEVATION AND PLAN OF BLAW-KNOX DEDUSTING UNIT.

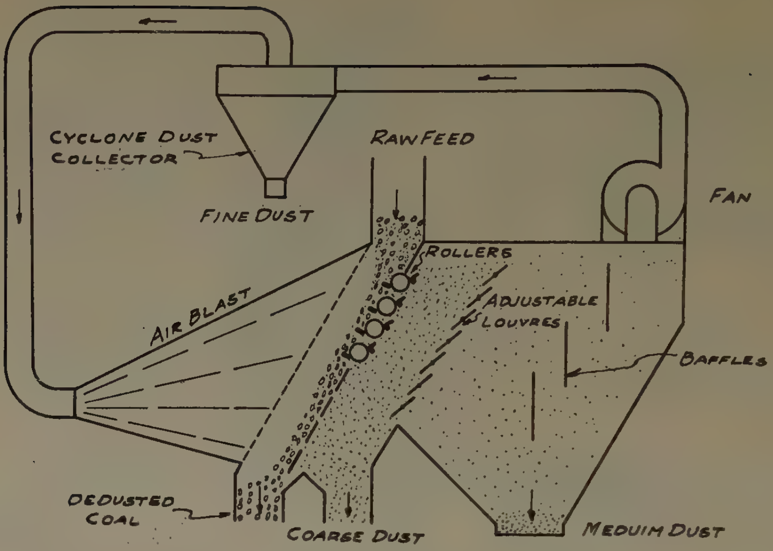

FIG. 10.—HUNTER DEDUSTING SYSTEM.

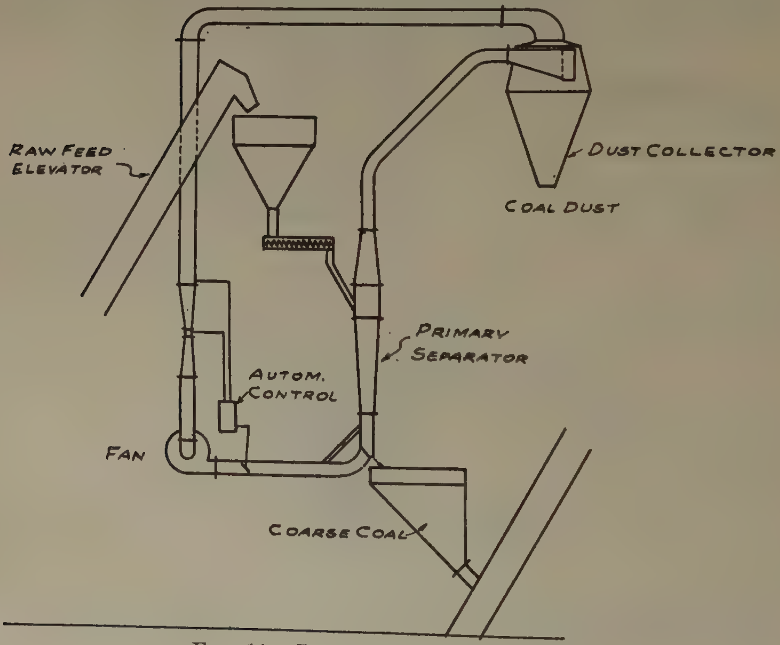

FIG. 11.—LESSING DEDUSTING SYSTEM.

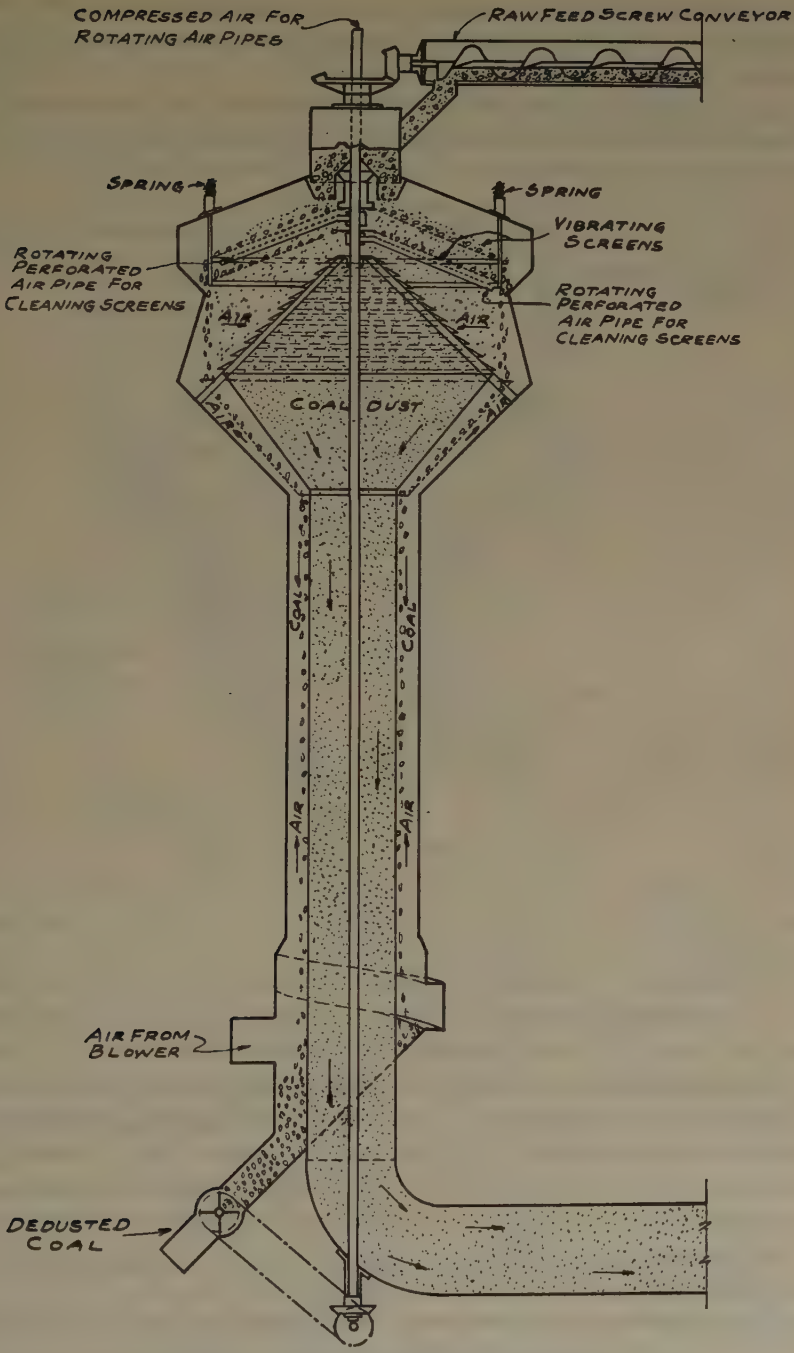

FIG. 12.—ALGAR DEDUSTING SYSTEM.

with compressed-air nozzles direct impinging jets that clear any screen openings that may have become blinded. This action is in a manner similar to that of the well-known soot-blower.

Fig. 13 shown by Chapman,[7] shows equipment somewhat similar in principle to that last described, except that the fan is contained in the housing of the equipment.

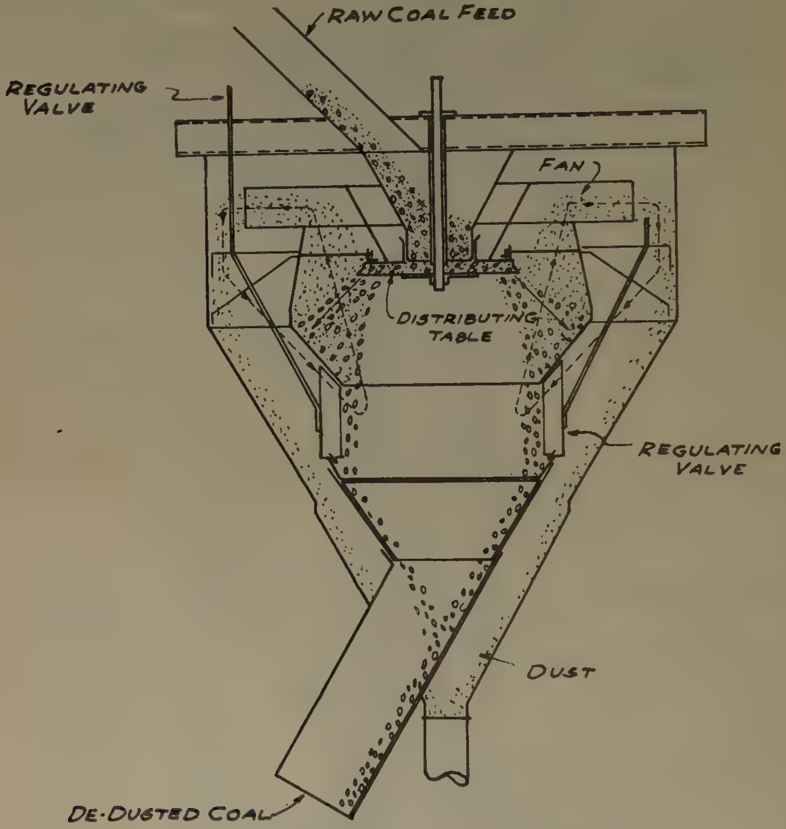

FIG. 13.—PFEIFER CENTRIFUGAL DEDUSTER.

One of the simplest and most efficient aspiratory equipments for coal is shown in Fig. 14. It is known as the Birtley aspirator and is the result of continued experimentation with dedusters of various types. The Birtley company has spent many years testing and developing this unit, as is borne out in Appleyard's papers.[8] This system depends on the suction of a fan drawing in air through an adjustable inlet nozzle, which lifts the coal off a stationary plate. A secondary adjustable air inlet is provided as shown.

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[7] W. R. Chapman: Recent Progress in Coal Cleaning Practices in Great Britain. International Bituminous Coal Conference, 1931.

[8] K. C. Appleyard: Dry Cleaning of Coal in England. *Trans.* A.I.M.E. (1931) **94**, 235.

K. C. Appleyard and E. O'Toole: Present State of Development of the Pneumatic Process for Dry Cleaning Coal. *Trans.* Amer. Iron and Steel Inst. (1930).

The coarser dust, after being lifted over a baffle, separates from the finer material and is deposited in a hopper that empties automatically whenever sufficient material accumulates to overcome a gravity-controlled sealing door. The fine dust continues on through the fan to a dust-collecting system. The Birtley company has installed many of these units and the tests given later show the results obtained.

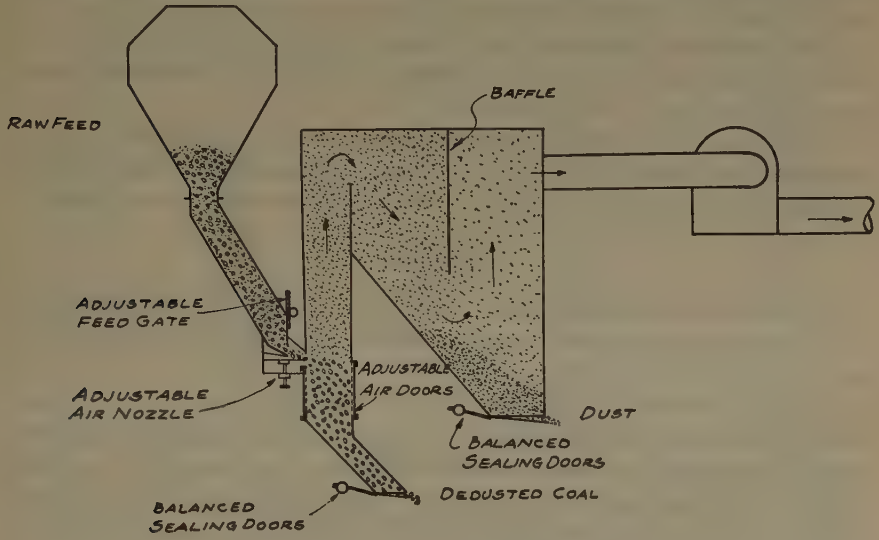

FIG. 14.—BIRTLEY DEDUSTING SYSTEM.

The Frankfort dedusting system (Fig. 11) is an American design embodying both screening and aspiration. This system uses a Hum-mer screen and two fans, one for providing two blasts through adjustable air nozzles and the other for drawing off the dust-laden air. This installation is very flexible and with it practically any size of dust may be obtained.

Other systems, depending, however, on the same principles, are the Daqua, Rema, Babcock, Raymond, Hildebrandt and Humboldt. All these units are illustrated in an article by Professor Rosin.[9]

DEDUSTING PRACTICE

Screening

In separating dust from coal, screening is often used without aspiration. With fine sizes moisture in the coal presents difficulties. In fact, Minikin states[10] that when natural moisture exceeds 4 per cent screening efficiency becomes prohibitively low. It is said that when the moisture is not more than 3 per cent the minus $\frac{1}{32}$-in. coal can be removed by screens of the Humboldt type. The screens, however, are very lightly loaded, only 4 tons per hour being fed to a unit having an area of 16 sq. ft. The manufacturer rarely quotes the capacity of a screen of this kind in tons per square foot of area. In lieu of such a declaration, he takes a sample and subjects it to a screen analysis, ascertaining the percentage of the various sizes down to dust. He then recommends the best mesh and weave of screen cloth, taking into account the screen sizes, moisture, shape of particles and stickiness.

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[9] G. P. Rosin: Wind Sifters. *Glückauf* (June, 1932).

[10] Reference of footnote 3.

Only screens of high-speed oscillating type will keep the screen deck in constant agitation and prevent blinding, though any of the electro-vibratory types will also give satisfaction.

It has been found that the efficiency of screening even under the most favorable conditions of feed, moisture and material is low when a sharp separation is necessary. For this reason aspiration has been adopted.

Aspiration

According to Braham,[11] dust is composed of materials ranging in size from macroscopic particles down to infinitesimal particles, which in the aggregate are known as smoke.

It can be divided roughly into three different classes—dust particles, cloud particles and smoke particles—and as these particles in their small sizes get cumbersome when expressed in fractions or decimals of an inch, Table 3 has been included giving various measurements in comparison with the micron. It should be remembered that one micron is equal to 0.001 mm. or 0.0000394 inches.

Dust larger than 10 microns allowed to settle in still air will fall initially with increasing velocity, while dust particles ranging from 10 microns to $\frac{1}{10}$ micron in diameter will settle with uniform velocity. Material of these sizes do not diffuse in air. Particles below $\frac{1}{10}$ micron and approaching $\frac{1}{1000}$ microns are of the order of smoke particles or fumes and cannot be settled in still air because they will readily diffuse. The latter particles are in active Brownian movement.

In dedusting, the great proportion of dust is above 10 microns and a percentage of the remainder is between 10 microns and $\frac{1}{10}$ micron. Many formulas are available for the calculation of the velocity of fall of dust particles and the work of Audibert, Gibbs and Blizard[12] should be consulted if the reader desires to follow this subject. Particles falling in a viscous medium such as air do not accelerate continually under the force

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[11] J. E. Braham: Dust Collection and Conveying Co., Cleveland, Ohio.

[12] R. Audibert: *Revue de l'Industrie Minerale* (1924).
W. G. Gibb: The Dust Hazard in Industry.
J. Blizard: *Jnl.* Franklin Inst. (1924).

TABLE 3.—*Comparison of Various Measurements*

Microns (Dia.)	Millimeters (Dia.)	Mesh (U. S. Std.)	Inches (Fraction)	Inches (Decimal)
1	0.001		1/25400	0.0000394
2	0.002		1/12700	0.0000788
5	0.005		1/5080	0.0001966
10	0.01		1/2540	0.000393
20	0.02		1/1270	0.000786
30	0.03		1/847	0.00118
40	0.04	325	1/635	0.001575
50	0.05	270	1/508	0.00197
60	0.06	250	1/423	0.00236
74	0.074	200	1/346	0.00289
80	0.08	170	1/317	0.00315
100	0.10	150	1/254	0.00394
147	0.147	100	1/173	0.00578
200	0.20	65	1/127	0.00788
250	0.25	60	1/101.5	0.00985
300	0.30	48	1/84.6	0.0118
350	0.35	42	1/72.6	0.0137
400	0.40	35	1/63.5(1/64)	0.0157
450	0.45		1/51.4	0.0194
500	0.50	32	1/50.8	0.0197
833	0.833	20	1/30.5(1/32)	0.0327
1000	1.00	16	1/25.4	0.0394
1168	1.168	14	1/21.7(3/64)	0.046
1651	1.651	10	1/15.4(1/16)	0.065

of gravity, but reach an equilibrium or terminal speed which depends on their density, diameter and the viscosity and density of the gas. If the vertical velocity of the air or gases equals the terminal velocity of the particle, the particle will remain in suspension.

The formula used in this case and the one from which the graph shown in Fig. 4 was drawn is as follows:

Let D = density of dust in grams per cubic centimeter (coal taken at 1.30 sp. gr.)
R = radius of particle, centimeters
V = terminal velocity centimeters per second.
Then $V = (12DR^2)(10^5)$.

Other things being equal, the velocity is proportional to the square of the diameter of the dust particle.

It will be seen, after studying this curve, that the results obtained from screening are essentially different from those afforded by aspiration. In the latter, the velocity of the air can be controlled to lift the desired size out of the coal and leave the coarser material behind. It is the ability of an aspirator to perform such work that marks its efficiency, and to properly design the unit, the screening test of the small sizes must be closely studied.

Apart from the screen analysis, other factors have a distinct bearing on the terminal velocity of particles; namely, the dust characteristics and the relative humidity of the gases. Dust may be spherical, globular or flaky and flocculent. Moreover, hygrometric conditions of the gases may cause trouble unless insulation or heat is provided. This involves a study of the relative humidity, dew point, and the permissible temperature drop in the system.

A second question that arises is the "grain loading"; that is, how much dust is being carried per unit volume of air. In general, the quantity is stated as "grains (avoir) per cubic foot" at a given temperature and pressure.

This figure varies greatly, depending on the class of work being performed. Table 4 gives the range of weight of material carried per cubic foot of gas. Where dust-laden gas is drawn from equipment whose function is other than the removal of dust from a certain product, it will be noted that the grain loading per cubic foot of gas is low. For instance, the gases coming from rotary dryers or kilns have as their primary duty the removal of moisture or the calcining of a product, therefore the grain loading in these cases is no criterion of the grain loading under dedusting conditions. In other cases, where coal such as 1½ to

TABLE 4.—*Weight of Material Carried per Cubic Foot of Gas*

Material	Gas Medium	Grains per Cu. Ft.	Remarks	Authority
Coal dust...............	Hot air	4.3	From dryers	
Zinc oxide...............	Gas	4.46		
Coal dust...............	Hot air	4.6	From dryer	
Coke and ore dust........	Hot gas	4.0–8.0	From blast furnace	
Dry cement............	Hot air	6.0–20.0	From rotary kiln	
Coal dust...............	Hot air	6.5–22.0	From dryer	
Coal dust...............	Hot air	8.9	From dryer	
Cement dust............	Hot air	9.25	From rotary kiln	
Coal dust...............	Air	50	Air suction system	
Coal dust...............	Air	70	Pneumatic conveyor	Holbeck
Coal dust...............	Air	66–131	Dedusting equipment	K.C.A.
Coal dust...............	Air	111	Dedusting equipment	B.K.Co.
Coal dust...............	Air	117–140	Pneumatic conveyor	Holbeck
Coal dust...............	Air	146	Dedusting equipment	H.F.H.
Coal.....................	Air	342–428	High pressure	
Coal dust...............	Air	374	High pressure	Quigley
Coal.....................	Air	1568–3730	High pressure	
Coal.....................	Air	3500	High pressure	

0 in. is being transported pneumatically and the air pressure is high, the grain loading per cubic foot is heavy.

In other instances listed in the table, coal dust is being carried by air either from pulverized-fuel grinding mills, dry coal-cleaning tables or dedusting units. They show a grain loading varying from 50 to 146 grains per cubic foot.

Proportioning a dedusting installation, therefore, should take into account the following factors: (1) the screen analysis, which indicates the quantity of dust to be removed; (2) the smallest grain to be left in the oversize, which determines the velocity for dust removal. With these figures and a consideration of the condition of the particles themselves as previously stated, the air volume and grain loading can be calculated and the apparatus proportioned.

In transporting dust suspended in air, it is well to remember that if the pipe or duct is vertical the force of gravity is directly downward and in direct opposition to this current of air. If, however, a length of horizontal pipe is used, the current of air is horizontal and consequently at right angles to the vertical gravitational force. The resultant of the two forces will tend to deposit dust in the bottom of the duct. The ducts, therefore, should be as near vertical as possible, and not less than 60° from the horizontal.

As a general rule, the feed to the aspirators proper is approximately minus ¼-in. material. If a screen is placed ahead of the aspirator or is incorporated in it, the screen size will be such that a minus ¼-in. product is available for dedusting by air.

Coal with moisture content of less than 5 or 6 per cent will be aspirated without undue trouble, and coals with even higher moisture contents have been handled, but of course "throughput" and efficiency are lower.

RESULTS

The Frankfort dedusting apparatus (Fig. 15) recently was tested by the author. This unit had been installed to supply the commercial market. The coal being prepared is a high-grade free-burning bituminous fuel, largely used for industrial and domestic purposes. As shown in Fig. 3, the ash content has a distinct tendency to increase with the decrease in size. As already stated, a growing fuel demand has to be met in semi-commercial and domestic boilers that are equipped with small underfeed stokers.

An underfeed stoker, one in which the green fuel is pushed up through the incandescent zone from the bottom of the bed, is best suited for burning strongly caking high-fusion-point ash coals, which require a "breaking" action in the furnace to prevent the formation of a crust over the fuel bed. The coal in question, when burned on such a grate, had its low-fusion-point ash lifted into the incandescent zone, and the high-ash dust either slagged or attached itself to the boiler tubes. It was also found that the presence of "fines" in the coal increased the draft resistance in certain areas, causing "blowholes" in the fuel bed. Extensive boiler tests were conducted and finally two products were decided upon to meet the demand.

For stokers installed on domestic loads such as apartment buildings, $\frac{5}{16}$-in. to 10-mesh coal gave the best results. It also had the advantage of being practically dustless and therefore clean to handle. For commercial stokers it was found that a percentage of the 10 to 48-mesh should be included in the larger size to insure the best results. The sizes decided upon were therefore:

Domestic stokers: $\frac{5}{16}$-in. to 10-mesh.

Semicommercial stokers: $\frac{5}{16}$-in. to 10-mesh plus 50 per cent of the original 10 to 48-mesh and as free from minus 48-mesh as possible.

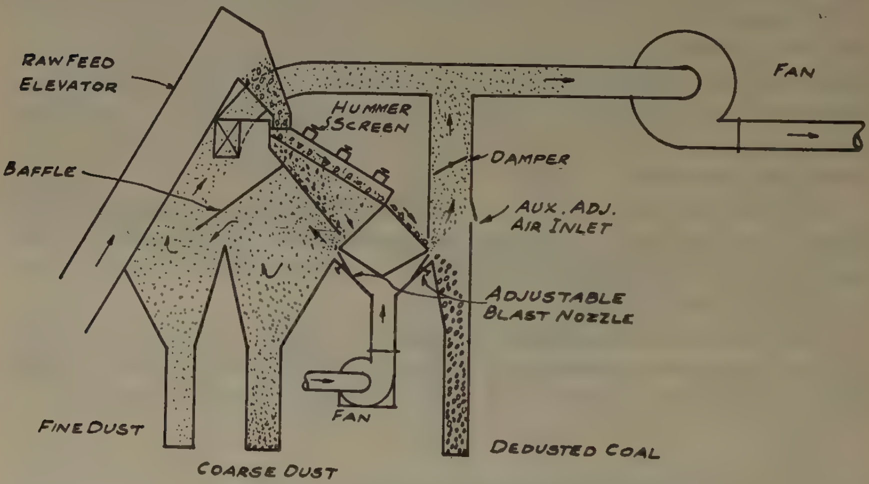

FIG. 15.—FRANKFORT DEDUSTING SYSTEM.

In the Frankfort apparatus (Fig. 15) the raw-feed elevator delivers 50 tons per hour of $\frac{5}{16}$-in. to 0 coal to a Hum-mer vibrating screen with a Ty-rod screen mesh having a free opening of 0.08 in. At present the screen is set at an angle of 35° from the horizontal, although provisions have been made for adjustment to any angle required. The feed coal, which contains approximately 9 per cent moisture, receives a preliminary separation into oversize and minus $\frac{1}{12}$-in. fines. Both products slide over plates and cascade over adjustable air gaps. At these gaps a positive air jet, delivering approximately 1400 cu. ft. per minute at 4.89 in. of water pressure and at a velocity pressure of 1.07 in. of water, impinges against the two cascading products and drives the fines out of the downward streams of coal. Under the present adjustments the $\frac{5}{16}$ to $\frac{1}{12}$-in.

material passes a nozzle 12 in. long by 3/8 in. deep, while the fines (minus 1/2-in. by 0) pass over an orifice 24 in. wide by 5/8 in. deep.

The dust, having been driven out of the coal streams, comes under the influence of the suction fan, which removes the dust from the system. However, in traveling to the suction fan the dust-laden air has its velocity lowered when passing through two settling chambers and a portion of the 10 to 48-mesh is settled out.

The adjustable damper and auxiliary air nozzle allows the velocity to be controlled. The suction fan handles approximately 6540 cu. ft. of dust-laden air per minute against a static pressure of 3.4 in. of water.

The equipment installed is as follows:

One raw-coal elevator: capacity, 50 tons per hour. Material 5/16 in. to 0.

One Hum-mer screen: capacity, 50 tons per hour. Size 4 by 8 ft. 1/2-in. Ty-rod screen; 3 V16 vibrators.

One pressure blower: capacity, 1410 cu. ft. per minute. Static pressure, 4.89; velocity pressure, 1.07 in. water.

One squirrel-cage motor: capacity, 5 hp. Three phase, 60 cycle, 220 volt, 1750 r.p.m., for the blower.

One suction fan: capacity, 6540 cu. ft. per minute. Static pressure, 3.4 in. water; 645 r.p.m.

One squirrel-cage motor: capacity, 10 hp. Three phase, 60 cycle, 220 volt, 860 r.p.m., for the suction fan.

At the present time the unit air requirements are as follows:

Total volume of air = 160 cu. ft. per min. per ton of coal treated per hour.

Length of nozzle per ton of coal treated per hour = 0.75 inches.

Air-nozzle velocity = 10,500 ft. per minute.

Horsepower per ton of fuel per hour = 0.20.

The Birtley deduster for normal practice generally requires the following:

Total volume of air = between 100 and 200 cu. ft. per min. per ton of coal treated per hour.

Length of nozzle per ton of coal treated per hour = 1.0 inches.

Horsepower per ton of fuel per hour = 0.18.

The Blaw-Knox system uses approximately 200 cu. ft. per ton per hour.

The curve in Fig. 16 was drawn for 5/16-in. to 10-mesh aspirated coal. It shows the percentage of plus 10-mesh product that was withdrawn with the dust by the blast and suction fans and it also shows the percentage of minus 10-mesh dust that has been entrapped in the larger sized aspirated coal. As a measure of efficiency, the shaded area below the curve, when divided by the area bounded by the 0 and 10-mesh ordinates. gives a fraction representing the efficiency.

TABLE 5.—*Screen Tests*

Size, In.	Mesh	Raw Coal		Aspirated Coal		Aspirated Coal		Dust	
		5/16-in.—0-in.		5/16-in.—10-mesh		5/16-in.—10-mesh 50 per cent of 10 —48-mesh			
		Per Cent	Cum. Per Cent	Per Cent	Cum. Per Cent	Per Cent	Cum. Per Cent	Per Cent	Cum. Per Cent
0.441		0							
0.371		Trace							
0.312	2.5	0.71		1.32	1.32	1.27	1.27		
0.263	3.0	3.52	4.23	3.92	5.24	3.37	4.64		
0.221									
0.185	4.0	15.29	19.52	25.17	30.41	19.33	23.97		
0.131	6.0	14.75	34.27	27.69	58.10	23.37	47.34		
0.093	8.0	12.95	47.22	22.71	80.81	21.44	67.78	0.08	0.08
0.065	10.0	10.57	57.79	11.01	91.82	11.90	80.68	0.46	0.54
0.046	14.0	8.05	65.84	4.03	95.85	6.75	87.43	1.42	1.96
0.033	20.0	7.19	73.03	1.72	97.57	4.13	91.56	3.51	5.47
0.023	28.0	5.85	78.88	0.91	98.48	2.86	94.42	6.05	11.52
0.016	35.0	4.83	83.71	0.55	99.03	1.99	96.41	9.40	20.92
0.012	48.0	3.75	87.46	0.29	99.32	1.17	97.58	9.91	30.83
0.0082	65.0	2.92	90.38	0.21	99.53	0.93	98.51	11.71	42.54
0.0058	100.0	2.00	92.38	0.11	99.64	0.54	99.05	12.33	54.87
0.0041	150.0	1.26	93.64	0.06	99.70	0.30	99.35	9.96	64.83
0.0029	200.0	0.89	94.53	0.06	99.76	0.17	99.52	9.17	74.00
0.0029	−200.0	5.47	100.00	0.24	100.00	0.48	100.00	26.00	100.00

RAW COAL

Size Mesh	Screen, Per Cent	Ash, Per Cent	Ash, Cum. Per Cent
5/16 in.—10 mesh	57.79	7.75	7.75
10 mesh—20 mesh	15.24	8.03	7.80
20 mesh—48 mesh	14.43	12.38	8.57
48 mesh—100 mesh	4.92	14.42	8.90
100 mesh—200 mesh	2.15	14.90	9.00
Minus 200 mesh	5.47	15.00	9.34
	100.00		

Some of the fines were carried over with the aspirated coal and some of the large sizes found their way into the dust, probably because of the percentage of moisture present and because of a rather large proportion of flaky material of small weight and large surface area which offered comparatively large wind resistance.

The curve in Fig. 17 represents the work of a Birtley unit which was installed for the extraction of the dust below 35 mesh. The graph has been plotted on the same basis as that in Fig. 16, and shows a high percentage of dust removed at the expense of a certain quantity of oversize. Had a 35-mesh screen been used, all the oversize would have been kept in the coal, but much more undersize would have been carried over.

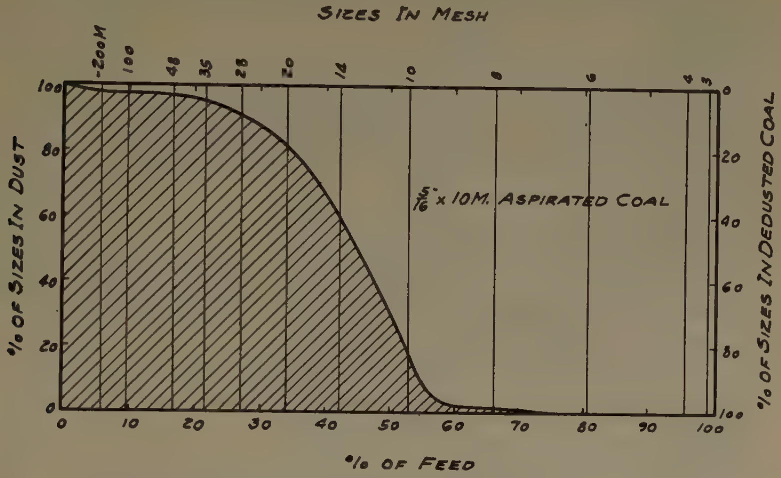

FIG. 16.—DEDUSTING PERFORMANCE OF FRANKFORT EQUIPMENT.

The curves in Fig. 18 represent tests of actual performances of Lessing dedusting equipment on three different coals. In curve A the performance is excellent, as the separation between ¼ in. to 50-mesh and minus 50-mesh is remarkably sharp. The separation between ¼ in. to 20 mesh and minus 20-mesh in curve B, while still very good, indicates that some of the minus material was carried over. Curve C illustrates the

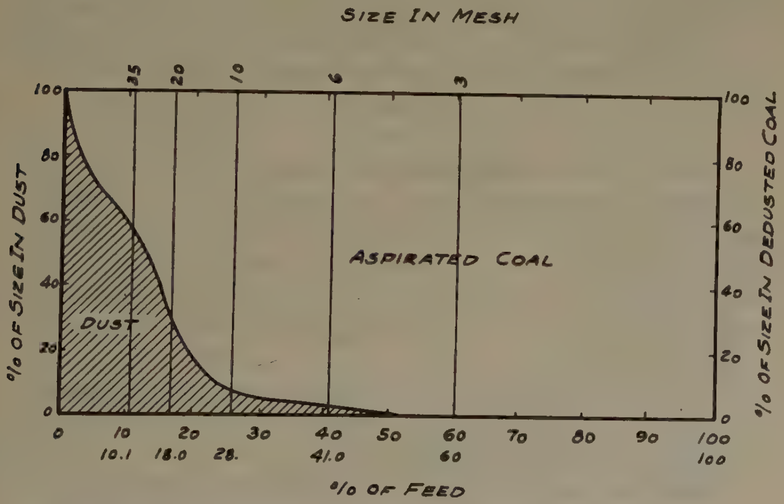

FIG. 17.—DEDUSTING PERFORMANCE OF BIRTLEY EQUIPMENT.

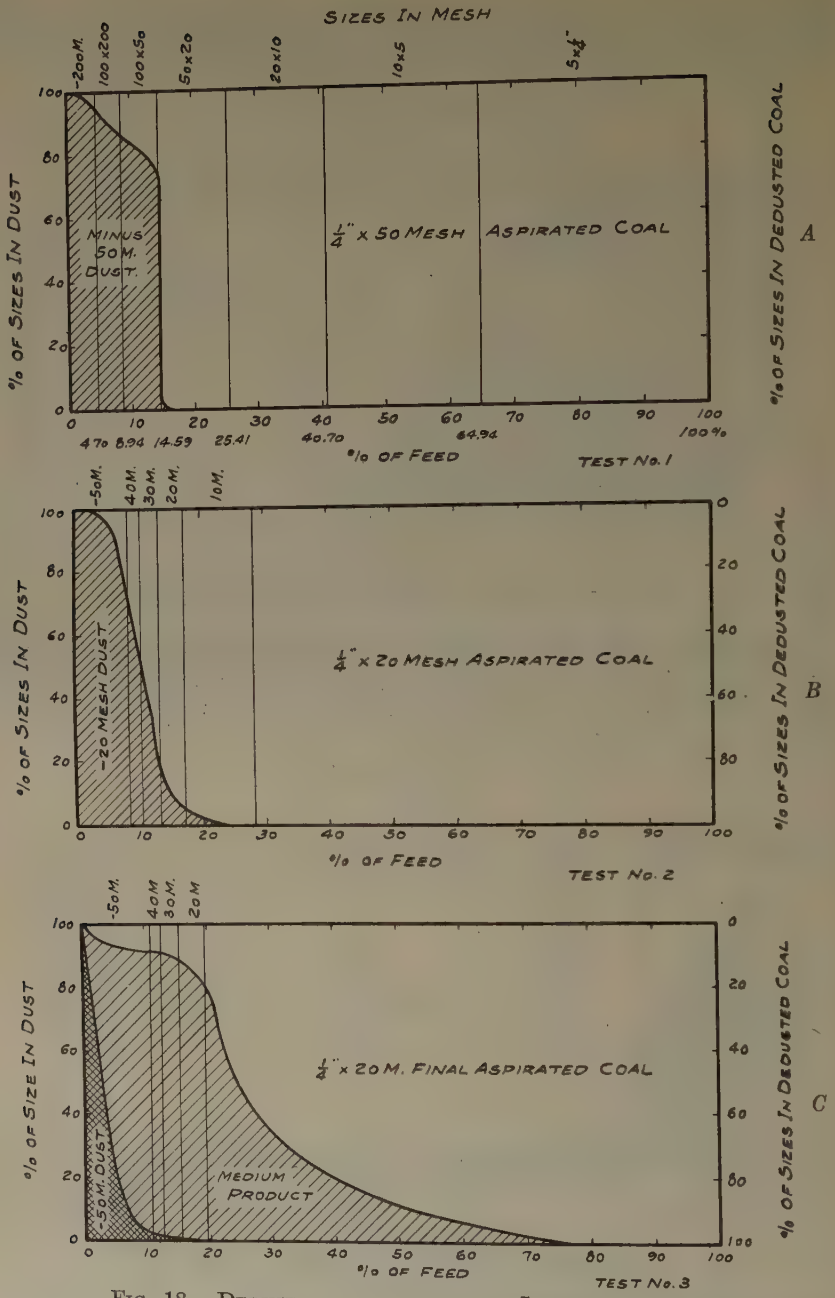

FIG. 18.—DEDUSTING PERFORMANCE OF LESSING EQUIPMENT.

results when making three products by aspiration. In this case the minus 20-mesh was first removed and then a further division made between the minus 50-mesh and the 20 to 50-mesh.

According to Chapman,[12] a dedusting apparatus similar to the type shown on Fig. 13 was tested under the conditions and with the results shown in Table 6.

Table 6.—*Test with Apparatus Similar to That Shown in Fig. 13*

Speed of distributing table, 250 r.p.m.
Diameter of apparatus, 9 ft. 0 in.
Horsepower required, 7.

	Test 1	Test 2
Size of feed to deduster..........	$\frac{3}{16}$—0 in.	$\frac{9}{32}$—0 in.
Rate of feed, tons per hour......	35	21.5
Weight of dust extracted, tons...	6	4.6

Screen analysis of dust:

	Per Cent		Per Cent
+1 mm.	4		1.4
−1 to 0.5 mm.	7	−1 by 0.3 mm.	10.3
−0.5 mm.	89	−0.3 mm.	88.3
	100		100.0

Dust Collection

Having separated the dust from the coal and transported it pneumatically where desired, it now becomes necessary to recover it. As already stated, the sizes of the particles vary from 20 or 48-mesh down to particles so fine that they approach the size of smoke or fume particles.

This product must be arrested and collected before it is discharged to the atmosphere; otherwise, a very serious dust nuisance will be created. Many types of equipment are available for this function, some of which are described later, but they depend mainly on the following principles:

1. Separation of particles by radial acceleration. Dust cyclone collectors are good examples of this principle. The sudden changing of direction of the dust-laden gases traps the dust particles in baffles or pockets, which deposit the dust by gravity into dust-collection hoppers that are out of the influence of the gas-stream flow. Such filters as the "Vorticose" and "Cindertrap" are typical examples of this system.

2. Passing of the gas stream through either cotton or woolen fabrics arranged either as tubes or on frames, thus allowing the dust to be filtered from the gas. This system embraces most of the bag house and allied systems using filter cloth.

Electrical precipitators such as the Cottrell, oiled or "sticky" filters and scrubbers using coke and water sprays will not be discussed here, as they are not applicable to the dedusting of coal.

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[12] Reference of footnote 7.

Sometimes the dust-laden air is sprinkled with finely atomized water to wet the dust and deposit it before the air escapes to the atmosphere (Figs. 19 and 20). Steam has also been tried for the same purpose.

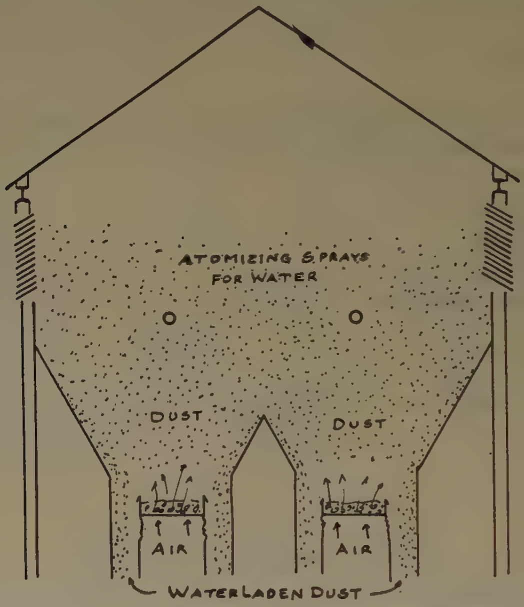

FIG. 19.—WATER-SPRAY DUST-SETTLING SYSTEM.

When dust is carried in a current of air at a velocity of 5000 ft. per minute practically no dust can settle.

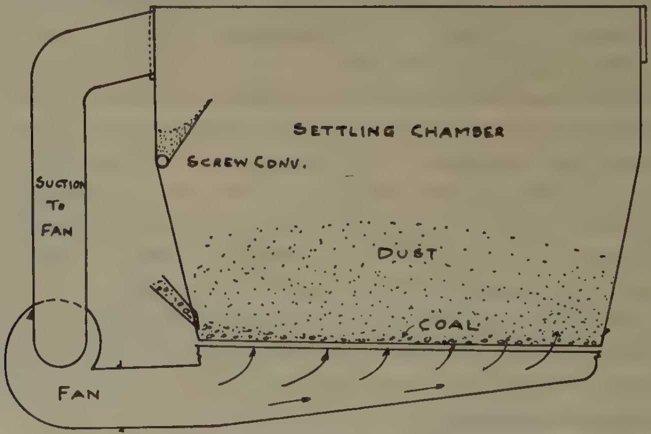

FIG. 20.—DUST-SETTLING CHAMBER SYSTEM.

A drop in velocity alone will not suffice for the deposition of dust, as the velocity should be well below 2 ft. per second if a fair efficiency is to be obtained. Therefore if this system is used in conjunction with a pneumatic cleaning plant or a dedusting plant in which large volumes of air must be used, the settling chamber must be gigantic, especially if the gases in the chamber suffer from eddies or turbulence.

The degree of removal of dust entrained in the gas depends on the time the dust is in the settling chamber, and remembering that the settling velocity varies as the square of the radius of the particle, the time required to deposit a dust having, say, 50 per cent of minus 300-mesh can be realized.

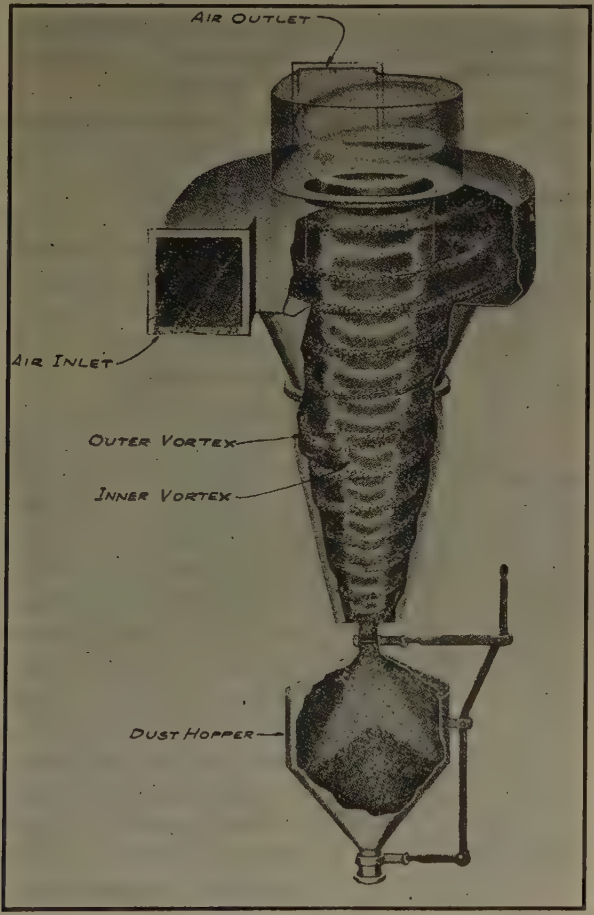

FIG. 21.—MODERN CYCLONE DUST FILTER.

The ordinary cyclone or centrifugal collector with an involute or tangential inlet is familiar to all. The dust-laden gas at a high velocity spirals downward, forming a vortex. After reaching the bottom of the collector, the gas turns and spirals upward to the outlet. The inner spiral forms a vortex of smaller radius. The path of the gas may be better understood by referring to Fig. 21. As a general rule, the ratio of the radius of the outer vortex to the inner is 2 to 1. In the selection of a cyclone collector, the size of the apparatus depends on the percentage of dust removal desired, therefore each collector should be designed to suit the installation it is to serve. Consequently, to design a plant, the volume of gas and the "grain loading," together with the moisture content and the temperature must be known. For any given dust condition any increase in the size of a collector over what is required will lower the efficiency.

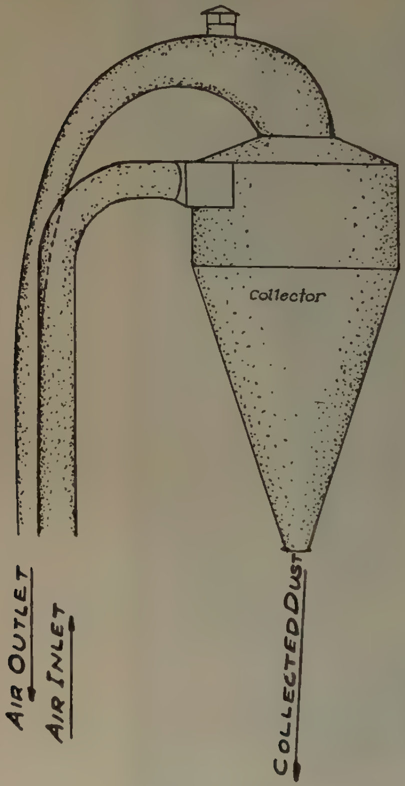

Fig. 22.—General type of cyclone collector.

It should be remembered that a high terminal velocity generally improves the efficiency of a centrifugal collector, but if after the unit is designed the volume to be cleaned is found to be greater than that for which the collector was planned, great resistance will be set up in the apparatus.

The radius should be designed large enough to treat the maximum gas volume. This requirement is in opposition to the maintenance of high velocity, so the problem becomes one of balancing the design. It has been stated that, if two collectors have the same gas inlet velocity, the greater the radius, the lower the efficiency. Or, having the same radius, the higher the velocity of the inlet gas, the higher the efficiency of separation, but the higher the power requirements. The efficiency also drops off rapidly, the finer the dust handled. Summarizing, it may be stated that this type of collector should be used for small volumes and dust particles of large size.

Impact filters or those which depend on the principle of abruptly changing the direction of the dust-carrying gases are illustrated in Figs. 23 and 24. These drawings show the method of driving the dust into dust pockets or traps, while allowing the gas to escape through ports and continue on through a succession of these traps until the outlet is reached. The dust so trapped drops vertically under the influence of gravity.

Cloth has been extensively used as a filtering medium where conditions will allow its installation. It cannot be installed where temperatures are high; in fact, for woolen materials and cotton materials the temperature should be kept below 250° F., and 210° F., respectively. The weave of the material should be properly selected, and if flannel is used the nap of the cloth should be arranged to be adjacent to the dust so that it will collect in the fine hairs projecting from the material. With this type of filter, the surfaces are generally arranged in one of two different ways. The frame type filter has the cloth attached to rectangular wooden frames, closely stacked in units, as shown in Fig. 26. Most of these units have automatic rapping or shaking devices, which remove the dust from the surface of the filters, dropping it into dust-collection hoppers below them. The other filter arrangement is the well-known tube or cloth bag, suspended from the top and so arranged that the gas must pass from the inside to the outside of the tube. With both styles of filters the fans handling the gases may deliver either a suction or a positive pressure, but a suction system has the advantage of making all leakage inwards, thus reducing the dust nuisance. When considering a filter of this type,

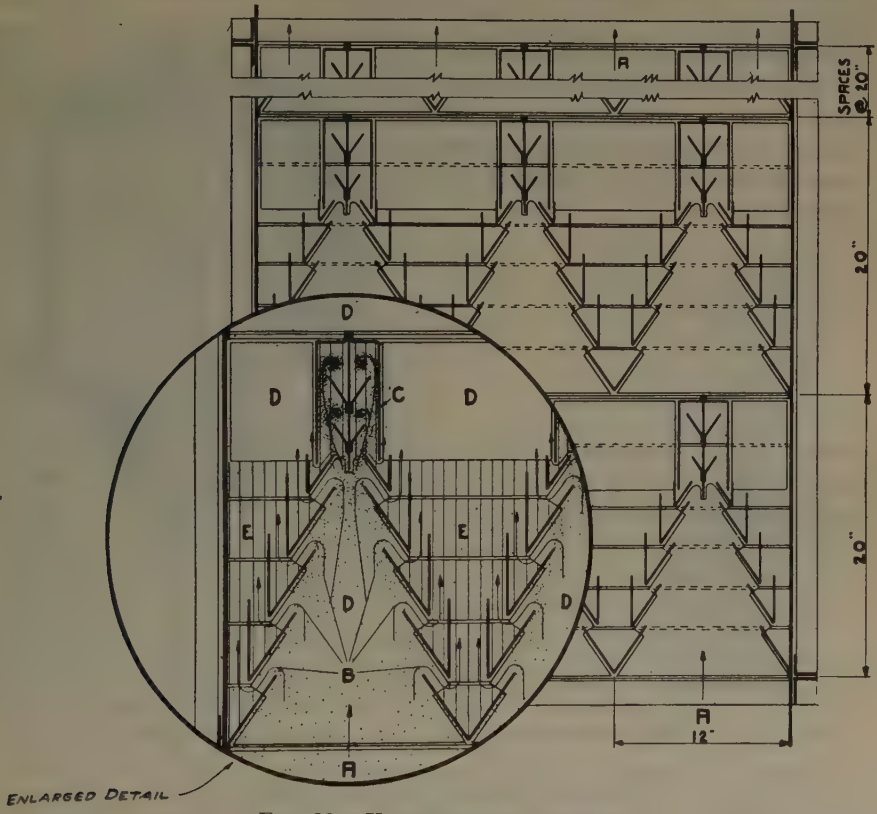

FIG. 23.—VORTICOSE DUST FILTER.

it must be borne in mind that the velocity must be so low that the static gas or air film that is against the surface of the filter will not be scrubbed off. If this happens, dust will be carried through the fabric and efficiency will be lost. Many velocities are used with filters of this class. The old-style bag houses had filter bags of 18-in. diameter, 30 ft. long, and allowed a velocity of from 0.4 to 1.0 ft. per minute through the fabric. Stated differently, a filter area was allowed of from 1 to 2.5 sq. ft. for every cubic foot of gas per minute. These units were high in cost and unhealthful to work in. The bags were shaken from 1 to 8 times per day and in one case the draft was 1.0 in. of water. With these units (used mostly at smelters) woolen cloth will resist the action of acid fumes better than cotton.

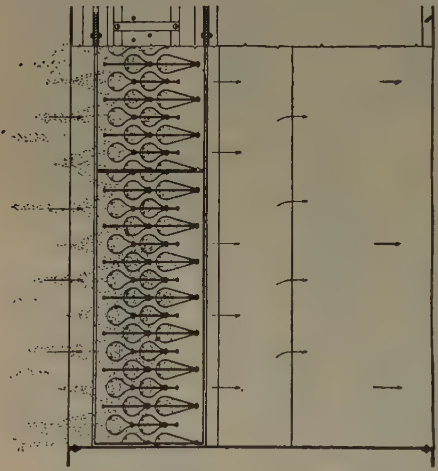
Fig. 24.—Cinder trap filter.

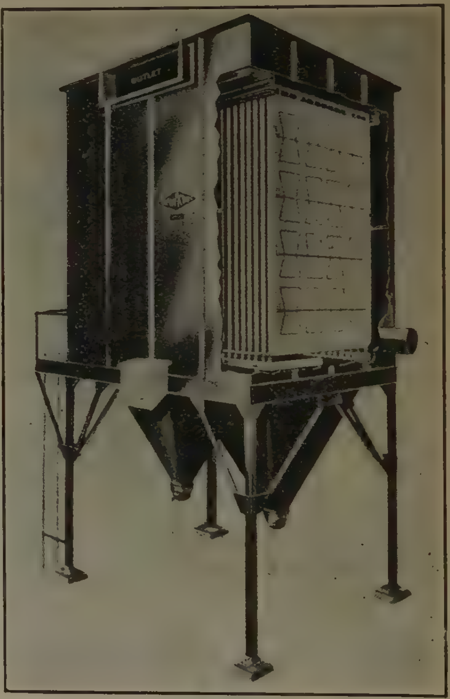
Fig. 25.—Blaw-Knox cloth filter.

Commencing with the above example, Table 7 gives a few of the velocities used with cloth filters in several different classes of work.

The grain loading per cubic foot has a direct bearing on the operation of the equipment, as this figure governs the frequency with which the bags or cloth must be rapped or shaken to free the fabric from accumulations of dust. It should be remembered that the resistance of the gases in passing through a cloth filter is comparatively high even when the cloth is new and clean. As dust is deposited on the surface of the fabric, the spaces between the threads clog and resistance increases.

Water gages for such filters have been known to be as high as 4 to 5 in., and it is easy to realize the power required to overcome this resist-

TABLE 7.—*Velocities with Cloth Filters*

Type of Filter	Material Filtered	Velocity through Fabric, Ft. per Min.	Remarks
Bag house	Gas and fumes	0.4 to 1.0	
Bag filter sec. type	Air and coal dust	1.18	
Frame type	Air and coal dust	1.25	
Bag house	Lead fumes	2.0	24 threads per inch warp and woof
Circular tubes	Air and coal dust	3.0	
Bag filters sec. type	Gas and flue dust	3.92	
Bag filters sec. type	Air and coal dust	3.0 to 4.0	
Frame type	Air and coal dust	4.0	
Bag filters sec. type	Lead fumes	4.37	
Bag filters sec. type	Zinc oxide	4.72	
Bag filters sec. type	Lead fumes	4.85	
Bag filters sec. type	Air and coal dust	4.0 to 8.0	Frequent automatic cleaning
Bag filters sec. type	Air and coal dust	8.0	
Bag filters sec. type	Air and coal dust	10.0	

ance. When the filter cloth is first installed, the nap is thick and "fuzzy" and strains the dust out easily, with a consequent high efficiency, but after continual rappings or shakings the fine hairs of the cloth break off and leave the threads bare, so that the filter passes dust through the fabric. Therefore, efficiency gradually declines unless the worn bags are replaced by new ones. The average efficiency over a period of years is about 90 per cent, although an efficiency of 99.5 per cent has been maintained in cases where sufficient care had been given to the equipment.

Attention must also be paid to the relative humidity of the air that is to be filtered. If the dust contains clay and the gases are cooled below the dew point, the filter cloth will become saturated with water and an impervious coating will be deposited on the fabric, which will destroy all filtering effect. In many places, especially in receiving gases from dryers where the percentage of moisture present and the temperature are high, a certain proportion of hot air from some outside source must be added to keep the gases above the dew point. Insulation of the filter casing also will aid appreciably in this respect. Where the dust is extremely dry a spark may fire the bag filters, a danger that can be reduced by sectionalizing the filter areas.

DESCRIPTION OF FILTERING EQUIPMENT

Figs. 19 and 20 illustrate the settling chamber. Fig. 19 shows water sprays, which bring down the dust before the gases are discharged. As already stated, the efficiency is poor unless the velocities are very low and eddy currents are avoided. Under such designs these requirements are practically impossible to attain, as there are always sufficient convection currents to hamper deposition.

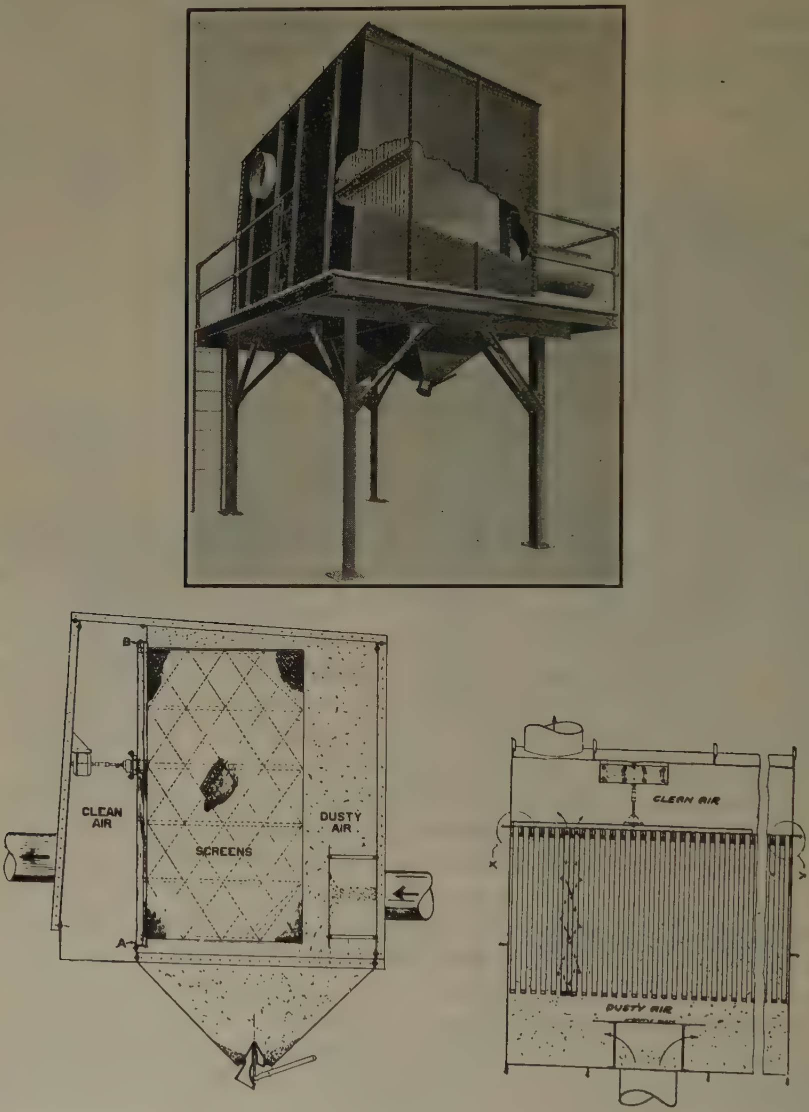

Fig. 26.—Whiting cloth filter.

Fig. 27 shows a combination cyclone and bag filter which has been designed especially for pneumatic cleaning plants. The dust-laden air enters tangentially at the top of the cone and, as the outside vortex spirals down, it deposits the larger dust in the dust-collection pocket.

The air receives its final filtration through the bag filters located above the cyclone. As this unit is under suction, no dust can leak into the atmosphere. When the bags need shaking, a valve in the suction outlet is opened to the atmosphere, and the inrush of air at atmospheric pressure quickly collapses them, thus freeing the accumulated dust. In recent installations the outside casings have been constructed of reinforced concrete, which affords excellent construction. This type of filter or dust collector is used by the Birtley company for all its coal-cleaning and dedusting installations.

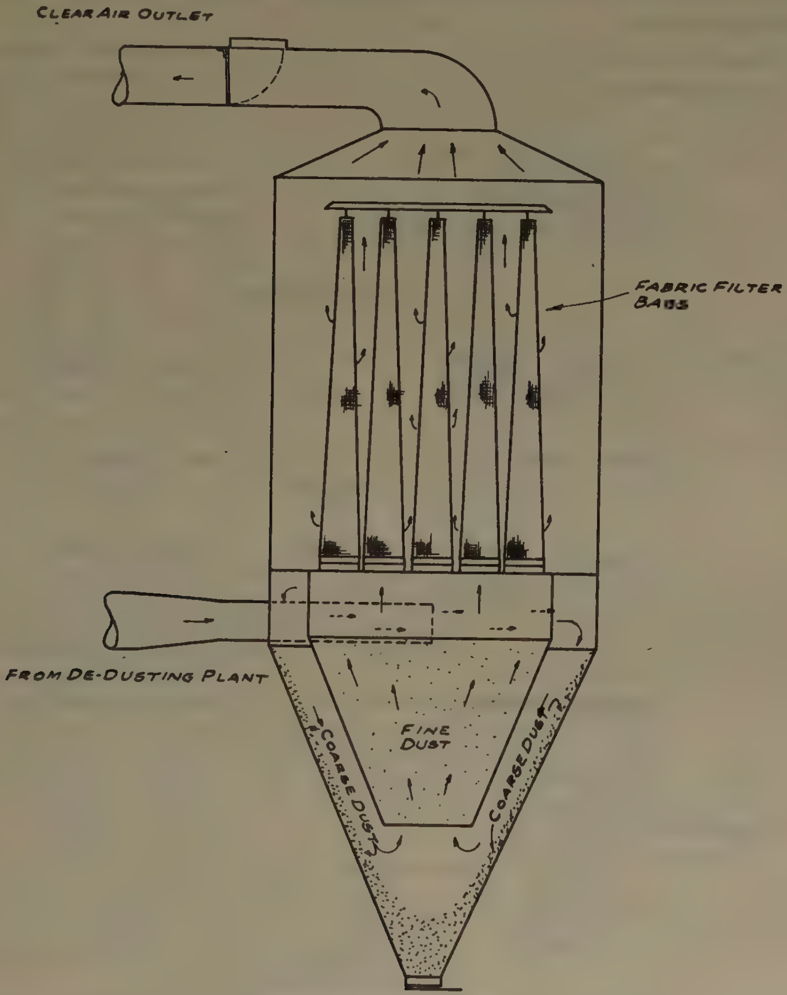

FIG. 27.—WARING COMBINATION CYCLONE AND BAG FILTER.

An American counterpart[13] of the above system is shown in Fig. 28. This unit is so arranged that the bags may be shaken at regular periods, varying from two minutes to two hours. The shaking period is approximately four seconds.

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[13] Reference of footnote 11.

The frame-type filters shown in Figs. 25 and 26 are self-explanatory. They have the advantage of being easily withdrawn from their frame supports.

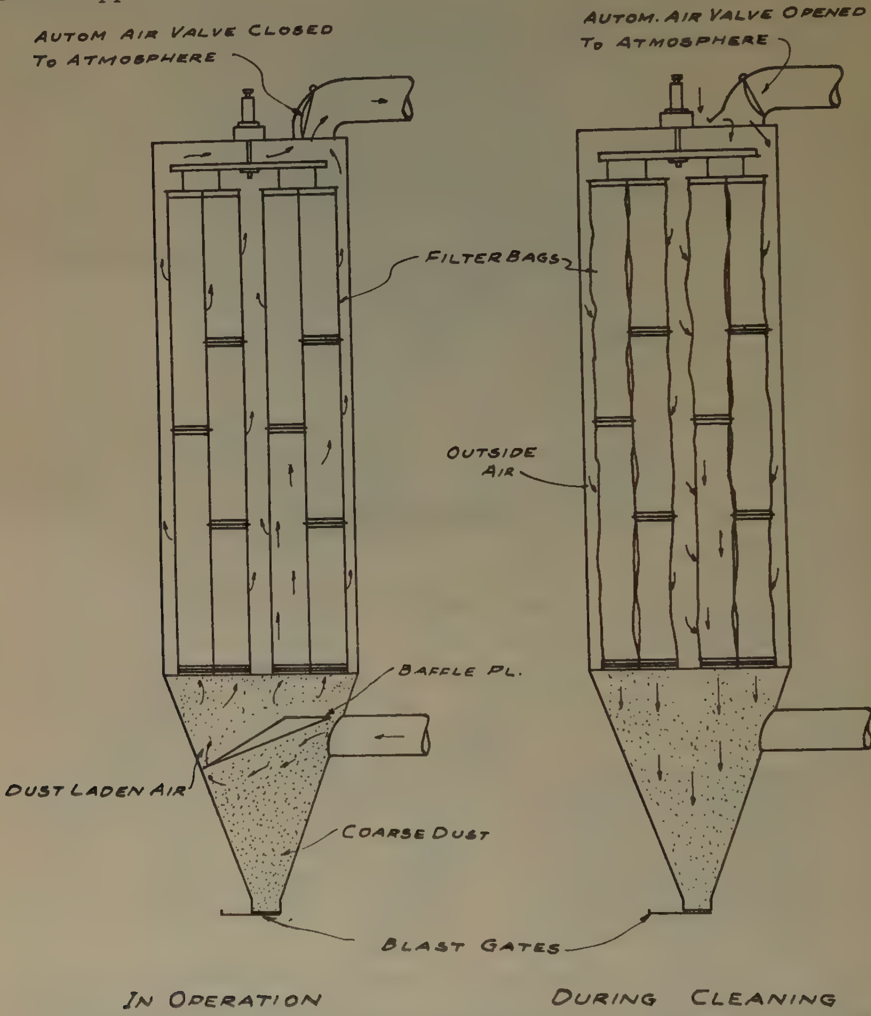

FIG. 28.—DRACCO COMBINATION CYCLONE AND BAG FILTER.

Impact filters have already been described and no further comment will be made.

RESULTS

As filters are used for many purposes and have to deal with many different gases, temperatures and filtrates, a table of results obtained from various sources is given in Table 8.

TABLE 8.—*Results with Different Filters*

Material Handled	Filter Area, Sq. Ft.	Fan Suction, Inches Water	Bag Resistance, Inches Water	Gas Volume, Cu. Ft. per Min.	Recovery, Lb. per Min.	Temp. through Filter, Deg. F.	Remarks
Zinc oxide	190	5	3	9,000	6.7		Fuming furnaces
Coal dust	1,300	6.5	4.6	10,050	12.7	142	From rotary dryer
Coal dust		3.7	2.7	7,650	4.17	116	From rotary dryer
Lead fumes	3,300			16,000		200	Fuming furnaces
Coal dust	972			1,150	0.75		From rotary dryer
Lead fumes	1,830			8,000		200	Fuming furnaces
Clinker and ash	17,833			70,000		220	Incinerators

ESTIMATES AND COSTS

To furnish some indication of the probable capacities of dedusting plants and the costs of operation and installation, the following figures have been deduced.

Taking the average of five Illinois mines and basing the calculations on a 300-ton per hour 6 to 0-in. feed to the tipple, it was found that:

Percentage of ⅜ to 0 in. in 6 to 0 in. = 19.0
⅜ to 0 in. raw feed to deduster = 57 tons per hour
Average percentage of minus 20 mesh to be removed = 23.5
Corresponding minus 20 mesh to be removed = 13.4 tons per hour
Average air consumption per ton of feed per hour = 360 cu. ft. per minute
Average grain loading = 150 grains per cu. ft.

Based on removing only the minus 48 mesh from the feed:

Average percentage of minus 48 mesh to be removed = 10.5
Corresponding minus 48 mesh to be removed = 5.89 tons per hour

Average air consumption per ton of feed per hour for different grain loadings:

70 grains per cu. ft.	340 cu. ft. per minute
100 grains per cu. ft.	240 cu. ft. per minute
125 grains per cu. ft.	174 cu. ft. per minute

It is interesting to note that Blaw-Knox in one installation has a grain loading of 111 grains per cubic foot.

COSTS OF INSTALLATION

The cost of installation per ton of feed to the dedusting unit varies with the throughput per hour.

For a plant of 50 to 100 tons capacity per hour and capable of removing minus 20-mesh material, a deduster complete with bag filters but not including the conveyors to and from the unit costs approximately .... $185 per ton of feed.

For a plant of 50 to 100 tons capacity per hour [but] removing the minus 48 mesh, the cost is approximately	$131 per ton of feed.
For a plant of 300 tons capacity per hour, the cost is about	$104 per ton of feed.
For a plant of 500 tons capacity per hour, the cost is about	$ 73 per ton of feed.

OPERATING COSTS

In estimating the operating costs of units, the following assumptions have been made:

Fixed charges—interest on investment, 6 per cent

Depreciation amortized at 3 per cent compounded: semiannually over 10 years; semiannually over 12 years; semiannually over 14 years

Maintenance, 5 per cent

Power at 1.25 cents per Kilowatt-hour

Hours per day, 10

Labor one man at 50 cents per hour.

The depreciation has been calculated on three different lives, namely:

10 years for a plant having 250 working days per year
12 years for a plant having 200 working days per year
14 years for a plant having 150 working days per year

Then for 57 tons per hour with minus 20-mesh removed:

	250 DAYS	200 DAYS	150 DAYS
Cost per ton of feed to deduster	$0.0288	$0.0312	$0.035
For 57 tons per hour with the minus 48-mesh removed: cost per ton of feed to deduster	$0.027	$0.0295	$0.0334
For 105 tons per hour	$0.020	$0.0212	$0.0228
For 300 tons per hour	$0.015	$0.0158	$0.0172
For 500 tons per hour	$0.0082	$0.0089	$0.0106

CONCLUSION

In conclusion it may be stated that the author has attempted to gather together all the information he could find pertaining to this rather obscure subject, adding some of his own knowledge and experience. Wherever possible, he has tried to give full credit to all sources from which the outside information was obtained. It is hoped that after analysis, should a coal illustrate the necessity of dedusting, these few notes will prove of some help. Due acknowledgment must be made to Mr. John Listak for the care he has shown in making the drawings for this paper.

DISCUSSION

(*Eli T. Conner presiding*)

K. C. APPLEYARD,* Birtley, England.—I have not actually seen the dust cars that Mr. Hebley showed on the screen, but I have had some experience of carrying dust in tank cars, both circular and square.

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* Managing Director, Birtley Co., Ltd.

In an ordinary open-topped car filled with coal dust I have known the dust to pack down so solidly in 25 to 50 miles of running that it would hardly take the impression of a footmark, and could not be discharged either by suction or in any other way than by shovels.

I have noticed also in Germany the difficulties experienced in carrying coal dust with square tanks, even when these are tapered towards the top, and I believe it is now common practice to pull chains through these tanks during the act of lifting them on to the tops of bunkers, so that the packed dust inside may be broken up and discharged readily.

The most satisfactory cars I have seen are the circular ones used in Germany, in which the discharge is made by compressed air and jets of air having a tangential cutting action round the coned bottom of the tanks, thus allowing a continual discharge.

Mr. Hebley shows a drag conveyor at the bottom of the car. This may operate satisfactorily and it may not. It depends largely upon moisture content, the amount of vibration that has taken place, with consequential packing in the dust, and I believe that the shape of the particles has a good deal of bearing on the matter.

Where dust particles are round, they do not possess that factor of tackiness that is possessed by dust particles of uneven shape which lock into each other, and my own feeling is that with a hoppered bottom and drag conveyor—with certain coal dusts, at any rate—the conveyor will merely pull its way through the dust, which will arch and prevent further discharge.

Two years ago, when speaking in Pittsburgh, I pointed out the necessity of finding some outlets for the use of coal dust and the great advantage to the industry that lies in the development of uses for this fuel.

I am a little sceptical about the field presented by briquetting, to which Mr. Hebley refers. These dusts are often exceedingly dirty and people will not buy dirty briquets at a good commercial price when they can buy clean coal for the same price or a little more. Another point is that in briquetting dusts the amount of binding material necessary is very large and the difficulties of adequately mixing the binding material with the fine dust are almost insuperable. Certainly I see no field for briquets in any country where there is a good supply of coal at a reasonable price.

In regard to dedusting as applied to wet washing, it is in my experience impossible to tell at present how much settling equipment can be eliminated by the use of a dedusting process. I also wonder very much whether the highest possible efficiency is always desirable. Any attempt to go up to 100 per cent of dust removal involves heavy expenditure, since it is always the last little bit that is difficult to get and costs most money. Like everything else, of course, it depends upon the circumstances of the particular case as to whether very high efficiency will pay for itself, but I often think that it does not always pay to assume that 100 per cent efficiency is the ideal; 85 or 90 per cent is sometimes more profitable from the commercial point of view.

The same thing, of course, applies to the subject of screening. To get 100 per cent screening efficiency costs a lot of money. Many people provide installations to give such a result and thereafter never use them at that high efficiency because it does not pay.

It seems to me so much more sensible to put in what is necessary for one's immediate requirements, plus the increased demands for efficiency that one can see are likely to come in the immediate years ahead, and let it go at that.

Coal cleaning itself is an outstanding example of the kind of thing I have mentioned. It always seems to be much better to clean coal to the extent it is required for one's market rather than to lose oneself in elaboration of plant to get high technical efficiencies which the market will not at present pay for and which are never used.

I have tried most of the English and German methods of dedusting. The cascade type, which has been very widely used, has not proved satisfactory. The dust—

especially if it has any moisture content—builds up on the louvres or the rolls, according to the type, with the result that the orifices vary in size and, as a rule, velocities increase, with the result that the fine dust is not always obtained but much more granular material is drawn out of the coal stream than is desirable.

Many systems circulate the air and only discharge a proportion of the air containing the heaviest dust. Most of them—after some hours of operation—reach a balance point, after which the efficiency drops off very considerably. By balance point I mean that the air becomes saturated with dust in circulation and becomes, in fact, a heavier fluid, and this tends to build up until the efficiency of the apparatus is destroyed, because the balance point is very high.

Tests carried out with systems containing rotating parts indicate that one rarely gets more than 75 per cent efficiency, owing to the fact that the breakage of particles keeps pace with the dust extraction beyond a certain point. This is clearly proved by taking the dedusted coal from such an apparatus, carefully registering its screening analysis and then passing it through the apparatus again, when the figures from the second dedusting will be found to be similar to the first dedusting, this being due to the fact that breakage is equal to the dust removal after some proportion like 75 or 80 per cent has been removed from the raw coal.

It may perhaps interest the members to have a note of some of the results obtained by dedusting in a large washery in England. With a raw coal containing rather over 20 per cent of $-\frac{1}{16}$ in. and just under 9 per cent $-\frac{1}{64}$ in., the dedusted coal going forward to the washer only averages a point or two either side of 0.5 per cent of material that will pass through a 60 I.M.M. screen.

The elimination of the slurry-forming materials due to this is very obvious in the washery working, since the percentage of solids in the wash water never rises over 4 to 5 per cent when the dedusting plant is at work, while if this is shut down the percentage of solids in the wash water is above 15 per cent in a very short time.

I think we are getting slowly to the stage when no cleaning plant dealing with small coals will be installed without adequate dedusting plant, and perhaps also to the stage when we shall be able to clean the dust satisfactorily and make some new use of it.

J. Griffen,* Pittsburgh, Pa. (written discussion).—In view of the rapidly increasing use of dedusting equipment in Europe as a preliminary to both the dry cleaning and washing of coal and the almost complete absence of such practice in the United States, this paper is of particular interest because it covers the subject so fully not only from a descriptive viewpoint but in giving many fundamental data.

It is true that in the cleaning of coal it is a general experience that little improvement is made in the finer sizes of raw coal which normally are removed by dedusting. This is particularly true of dry cleaning, and all carefully obtained test data I have seen covering such plants in the United States and Europe generally show negligible reductions in impurities in any dust sizes. The Birtley Co. has, perhaps, made greater progress in cleaning the finer sizes of coal than other manufacturers of dry-cleaning equipment. The gravity concentration of coal with water can effect an appreciable cleaning of certain dust sizes. In the Pittsburgh district the Rheolaveur installations cleaning metallurgical coal, at Banning and Warden washing plants of the Pittsburgh Coal Co., the Nemacolin plant of the Buckeye Coal Co., and the Clairton Washery of the Carnegie Steel Co., are washing the fine coal with a higher efficiency than is general. At these plants, a gratifying removal of fine refuse down to 100 mesh in size is accomplished. Similar performance is shown by the Rheolaveur plant of the American Smelting & Refining Co. at Cokedale, Colo. However, such results are attained only with great care in plant design and operation. It is probable that greater plant investment is required for thickeners and filters than would be the case if dedusting had been employed and the raw dust added to the metallurgical product. The improvement in ash and sulfur of the final metallurgical coal due to the cleaning effected on the dust may justify the additional plant cost. But when coals are cleaned for ordinary commercial use, the reduction in ash and sulfur obtained by washing the dust probably is not justified in all cases, particularly when the added moisture is considered.

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* Koppers-Rheolaveur Co.

As the results that can be obtained by dedusting are more fully appreciated, its wider use in the United States must follow, and Mr. Hebley's paper will undoubtedly stimulate interest in such possibilities. My visits to Europe in 1929 and 1932 gave opportunities to study a number of dedusting installations and the increased use of dedusting in connection with coal cleaning during the three-year interval was very striking. It would appear that much of the development of equipment has been along the lines of rule of thumb and I have been able to locate only one article that seriously studies the principles that must be utilized for efficient and practical operation. This paper[14] is referred to by Mr. Hebley but it seems to me it warrants wider attention.

After outlining the principles governing separation of fine material by air, the authors conclude: "the intended purpose of aspiration can be efficiently accomplished only when the air rises vertically and thus operates against the constant force of gravity acting on the solids. On this law are based the aspirators used in the laboratory. A large number of aspirators utilize other forces than gravity, especially centrifugal force. In these applications, the process is more along the line of a division where particles of all sizes are separated depending upon the location of the particle in the rotating air current. In other words, only with a vertical rising air current is it possible to make a theoretically perfect division provided all particles have the same specific gravity and the same shape. Where one resorts to additional forces other than gravity, the separation becomes theoretically more imperfect and when operating difficulties are considered the separation is still less efficient. The disadvantage of the use of horizontal air currents is that some of the finest particles in the lower layers begin to drop out according to the principles of the dust chamber; while the larger particles of the dust in the upper layers, which must be removed, cannot drop out unless the horizontal duct is quite long."

This indicates that an aspirator, to obtain the best efficiencies, should utilize the separating effect of a vertical air current acting in a passage with a uniform cross-section and over some considerable length. The length is necessary because of the practical impossibility of maintaining uniform velocity at all points of a given cross-sectional area.

My examination of the various aspirators already developed brings me to the conclusion that the Lessing & Birtley machines more nearly approach compliance with this requirement than any of the others. The Lessing is perhaps the most efficient aspirator developed but involves many practical limitations, particularly the additional mill head required and the possibilities for breakage. The present Birtley aspirator was developed to overcome the shortcomings, both theoretical and practical, of the earlier types and represents the results of some eight years of experience in this field. It appeals to me as combining fundamentally sound principles with provisions for minimizing practical operating difficulties such as the choking of the machine by damp or wet feed material. The fact that the feed rate is largely regulated by the primary air is an important feature in successfully handling damp materials.

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[14] G. P. Rosin and E. Rammler: Aspirators and their Investigation. *Glückauf* (June 11, 1932).

M. I. Dorfan, Pittsburgh, Pa.—Any comparative discussion of dedusting method efficiencies in order to have any meaning must have reference to the free moisture content of the material to be dedusted. In other words, any two tests for dedusting efficiency must be made on materials with a comparable moisture content or they cannot be compared. It is obvious that the difficulty of dedusting increases rapidly with an increase in surface moisture. Perhaps Mr. Hebley did not make this point clear and possibly did not even consider it in making his efficiency comparisons.

Another point that is extremely important in the design and consideration of dedusting methods is degradation. The degradation must be kept to a minimum. Two factors that will contribute to the elimination of degradation are: (1) The lack of moving parts, with their consequent mechanical abrasion; (2) the facility of lining the abrasive parts with some resilient nonabrasive material such as rubber or linatex.

Therefore a design with absolutely no moving parts and a construction lending itself to easy lining, such as the Blaw-Knox, is a step in the right direction.

Colonel Appleyard is of the opinion that, because of this degradation, it is not practical to make a second pass of the material in order to raise the efficiency of dedusting. This is probably true—but why not make the second pass immediately after the first, thus causing practically no additional degradation of material? In the Blaw-Knox deduster the material dedusted is immediately turned over and again subjected to an air blast, which gives double dedusting without additional degradation.

In regard to dust collectors in connection with dedusters we feel that Mr. Hebley is not quite up to date in his remarks. It is no longer the practice to use cloth such as flannel with a long soft nap. A hard woven material is now used. Consequently Mr. Hebley's statement that the efficiency of collectors declines with age, owing to the wearing out of the nap, is not true today.

It is also practical in these days to handle materials in gases up to 750° F., by using asbestos bags instead of wool and cotton.

H. F. Hebley (written discussion).—Referring to Colonel Appleyard's discussion regarding the possible market for briquets, the dust would necessarily have to be low in ash content. A great deal of work has been done on "briquetting," both with and without a binder. The possibility of using the dust in powdered-coal furnaces, without further pulverization, is receiving considerable attention, both in Germany and America, and the results obtained are very gratifying. The Uses of the Products from Dedusting Apparatus forms a subject for another paper.

Mr. Griffen draws attention to the use of a vertically rising current of air which gives a uniform separation of dust from the coarser sizes. In our experimental work we have found that efficient results can be obtained with the use of such equipment. The results obtained by Dr. Lessing in his method of dedusting are illustrative of the high efficiency obtained with a vertically rising current of air.

In reply to Mr. Dorfan's statement, it is well known that gases up to a temperature of 750° F. can be handled provided asbestos bags are used; but the use of air at a high temperature, while advantageous for drying the coal, introduces problems that more properly apply to the heat drying of coal. For instance, heating above a certain point will tend to drive off the lighter volatiles of the coal. Again, the specific volume of dry air at 750° F. is 31 cu. ft. per pound compared with 13.3 cu. ft. per pound at 70° F. With high-moisture coals, the use of heated gases will call for the analysis of such conditions as dew point, relative humidity, changes in specific volume, etc., which are beyond the scope of this paper.

H. Louis,* Newcastle on Tyne, England (written discussion).—Mr. Hebley's paper gives a very fair summary of the present knowledge of this subject, but adds little to it. I suppose it is hopeless today to attempt to protest against the hybrid word "dedusting," as it appears to have been universally adopted by mining people.

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* Secretary, North of England Institute of Mining and Mechanical Engineers.

I regret that Mr. Hebley gives no information as to the efficiency or otherwise of such substances as potato flour in settling clay, etc., suspended in the effluent wash water, a practice that is gaining ground on the Continent of Europe. I am inclined to think that Mr. Hebley has underestimated the difficulty of transporting coal dust. My experience is that when coal dust is conveyed any distance, especially by rail, the inevitable vibration shakes the dust into a practically solid mass, and makes the discharge very difficult.

The late R. D. Lunnon published a useful paper on this subject a few years ago.[15]

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[15] R. D. Lunnon: The Laws of Motion of Particles in a Fluid. *Trans.* Inst. Min. Engrs. (1928–29) **77,** 65.

Relationship of Ore Dressing and Coal Preparation

By E. A. Holbrook,* Pittsburgh, Pa.

(New York Meeting, February, 1933)

The art of ore dressing is as old as civilization itself. Jason's search for the golden fleece was perhaps only the use of sheepskins for catching gold from gold-bearing sands washed over them. From Hoover's translation of Agricola we gather that in the middle ages ore dressing had advanced through the stage of hand sorting to water concentration by jigging. German methods and apparatus continued in the lead until the closing years of the past century; indeed, it is little more than a generation ago that many ore-dressing plants in this country were designed and operated by German-trained ore-dressing specialists.

Meanwhile the development of the ore deposits in this country during the nineteenth century brought forth new inventions and methods. Blake's invention of the rock breaker in 1858 and the coming of the Wilfley table about 1896 are random examples of the improvements caused by our expanding needs. Yet ore dressing remained an art developed and practiced more or less by cut and try methods. The publication of the volumes on Ore Dressing by Dr. Robert H. Richards in 1903 marks the beginning of American ore-dressing practice as a science. Previously we had as guides only a few incomplete books based on foreign practice, or scattered references and articles in our own TRANSACTIONS. Dr. Richards set down all the developments that had gone before, and outlined the ways for a scientific approach to the subject. He gave us a starting point. Since then ore-dressing practice and technique in this country have set the pace for the world.

Coal preparation has been called "ore dressing upside down," because in ore dressing we generally save a small percentage of the heavy valuable mineral and discard the lighter gangue, whereas in coal preparation we save the larger quantity of light mineral and discard the smaller percentage of heavy waste. By comparison with ore dressing the history of coal preparation is brief, extending backwards hardly 100 years, for although coal had been dug and used locally for centuries it became an important article of commerce only with the coming of the industrial revolution 150 years ago. In new mines it was to be expected that the thickest and purest seams should be worked first. So long as these lasted there was little thought of coal preparation other than hand picking of the larger impurities and simple sizing to supply the real or fancied need of the consumer. I have read that one hundred or more years ago, in England, it was noticed that when impure coal was thrown into a brook, the slate would fall to the bottom and be caught among the stones, while the pure coal would float down and be recovered below. This led naturally to the building of the trough washer, a long wooden sluice with riffles on the bottom through which water ran and into which impure coal was shoveled. When the riffles were full of impurities the stream of water was diverted and the riffles cleaned. After all, this was only the principle that had been used for centuries by placer miners, and yet developed independently by the coal miner when need arose.

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By 1870, France, Germany and England were mining considerable coal that required cleaning, and then began an interrelated development in these countries of coal crushing, screening and cleaning or jigging machinery. The principles used had long been known in ore dressing. The difficulties of the coal-mining industry were to adapt the machinery so as to treat large quantities of mineral at a very low cost, for in general the product of a coal-cleaning plant is worth only a few cents a ton more than the untreated product from some competing district; whereas in ore dressing there is generally a wide margin in value between the untreated and the treated mineral, if indeed the untreated mineral has any value whatever. Ore dressing, therefore, developed through the spur of absolute necessity, over centuries of time, while coal preparation is a competitive refinement of a comparatively recent date.

Historically, it is reported that the first hand-operated coal-washing jig was used at the St. Etienne collieries in France in 1837. In 1848 Berard invented a mechanical piston jig that rapidly came into use in France, Belgium, Germany and England. A number of types of other washers were invented shortly thereafter and by 1860 the process of coal washing was established in Europe.

One of the first references to coal screening gives the following information for 1740: "In this year the mischievous practice of screening coals was first introduced at Willington colliery by Mr. William Brown."[1] Bar screens were introduced generally in Europe about 1860. Nevertheless, up to the year 1870, cleaning and sizing machinery were employed only when coke was made from otherwise dirty coals. Sloping bar screens, sizing trommels and traveling belts made their appearance about this time. Then came the introduction of the continuous coal jig with automatic removal of refuse. In 1873 the first coal-washing plant of modern type was built in Westphalia. In 1873 a plant of the Berard type was erected at Broad Top, Pa., although previously, in 1869, a simple coal-washing plant was constructed near Pittsburgh under the direction of a German engineer. Thus, although the United States was early in the field, little was done to improve the processes here, and the real developments of the next years were left to Germany. Among these developments were the Luhrig feldspar jig, methods of drying fine coal and conveying apparatus.

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[1] *Trans.* North of England Inst. Min. Eng. and Mech. Engrs. (1865–1866) **15**, 205.

By 1880 most of the essential features of the jig washeries had been introduced and later progress has been in the nature of refinements. Other methods of cleaning coal, as by pneumatic means, heavy solutions and flotation, are of recent development. In the past few years competitive conditions in America have made it necessary for us to develop methods of drying fine coal more advanced than European practice.

In the anthracite region of Pennsylvania, crude jigs were used between 1860 and 1870, although the first extensive washing of anthracite was probably that undertaken by the Lehigh Coal and Navigation Co. about the year 1875.

Anthracite coal preparation by sizing started before the Civil War, and was followed in the 70's and 80's by various jigs and cleaners, which developed into the great coal breakers of today. The high value and limited supply of the anthracite as compared with bituminous coal made these early developments logical. In the 80's a coal-washing plant was a novelty in the bituminous regions of the country. Indeed, up to the time of the World War, plants for washing bituminous coal were possible only where a special coal was to be used for a special purpose, or where the freight rate on a naturally pure coal was so high as to make feasible the washing of a local but more impure coal. The early development of bituminous-coal washing in America was largely in Alabama and certain western states, and was based on the need of making metallurgical coke from local naturally dirty coals. Washing and cleaning bituminous coal for the general commercial trade was a secondary and recent development.

The World War, with its subsequent high freight rates, did more to educate the coal consumer to the economic value of prepared bituminous coal than any other factor, and in the past 20 years there has been more progress in scientifically preparing coal than in all the preceding years. The past five years have been especially noteworthy. In general, this progress has followed the older brother "ore dressing" by adapting its principles to the dressing of a product where low cost, large tonnage, sizing, friability and market price were often more important factors than the scientific excellence of the process. For example, during the years from 1900 to 1910, the author visited many ore-dressing and coal-preparation plants. Nearly every ore-dressing plant was being operated under careful analytical and scientific control; in few coal-washing plants was the control more accurate than to judge the results by looking at the products. Today the modern coal-washing plants are designed scientifically and are under analytical control that compares well with that at ore-dressing plants, making allowance for the difference in the value of the material treated.

In comparing ore dressing with coal washing we must recognize the difference in the materials treated. The minerals and rocks of ore dressing are inorganic chemical compounds of fairly fixed composition. Coal is a complex organic chemical substance, containing an almost infinite number of compounds. It may be subjected to proximate, ultimate and physical analysis, none of which is entirely satisfactory for the work of preparation to bring out special properties of coal and the value of the coal for specific uses. In fact, today there is no acceptable standard as to what constitutes clean bituminous coal. The anthracite industry, however, does have a generally accepted standard of purity.

Generally in ore dressing there is a gangue of, say, 2.70 sp. gr. with a valuable mineral or minerals of definite and heavier specific gravity. With coal there is no fixed specific gravity either of the good coal itself or of the slate, shale and other refuse. Acceptable bituminous coal at the same mine may vary in specific gravity from 1.25 to 1.60, depending on size and on the inherent ash content of the individual lump, while the shale refuse may vary from 1.40 up to 2.65 sp. gr., depending on the coal content of the individual piece. An additional difficulty comes from the fact that the individual piece of coal or shale may be flat, which causes it, in the water or air-treatment apparatus, to act like a cubical piece of much lower specific gravity. With anthracite, the specific gravity may range from 1.40 to 1.75 while the slate refuse may run from this point up to 2.70. Pyrite, the heaviest impurity found in coal, may have a specific gravity up to 5.10.

A third important difference is that of size. With the majority of ore-dressing processes, the ore is broken or crushed to a very fine size before treatment is completed. With coal, size is most often the important factor of value. For example, an egg size of anthracite may be worth $7.50 per ton to the producer, while a buckwheat size may be worth only $1 per ton. With bituminous coal the larger sizes are generally the more valuable. Even where the coal is to be crushed later for stoker or coking purposes, it is important to wash the coal in the larger sizes, because of the difficulty of removing water from the fine sizes. In dry cleaning the production of dust from fine sizes creates a dust-handling problem that must be kept to a minimum. In the production, handling and crushing of coal preparatory to treatment, therefore, every known precaution should be taken to prevent the production of fines, and separation of coal from refuse should be made on the largest sizes possible, even at the cost of a small percentage of coal lost with the refuse. With these points in mind, let us compare coal breaking with ore breaking.

Breaking in the Mine

Ore is drilled, and then blasted by dynamite in the mine. The shattering power of dynamite helps the breaking process. Sledgehammers are kept handy to break pieces that will not go through the grizzly screen at the mine or mill. Woe to the mine boss and miner who loads rock too large for the crusher to handle. On the other hand, dynamite is a crime at the coal face. True, the mechanical coal cutter makes some fines or "bug dust," but it enables the rest of the cut to be broken gently with permissible powder with a minimum of fines. Experiments at one coal mine showed that the additional 15 per cent of coarse coal that could be secured by careful control of the amount of the explosive used made the difference between profit and loss at that mine.

Hand sorting of ore underground in the stopes is rather unusual, whereas in coal mining each car of coal coming out of the mine has attached the number tag of the miner who loaded it, and if the larger pieces of impurities have not been removed by the loader he may be "docked" or not paid for the car, or in some cases discharged. In many mines it is an axiom that coal cleaning must begin at the face underground. Any comparison of loading clean ore and clean coal must also recognize that the ore miner is generally paid by the shift and is a regular company employee, while the individual coal miner is paid by the number of cars or by the number of tons he loads, and in the anthracite mines is often himself an employer of assistants; really a small contractor.

A further comparison of ore and coal in underground preparation is in the care of handling the mined product. Broken ore may be shoveled and handled roughly, dumped down chutes and even reloaded several times before appearing at the mill. The hardness of the material minimizes breakage, but in general breakage here does no harm and saves work for the crushing machinery in the treatment plant. With coal, from the mine face to the consumer, every effort must be made to minimize breakage, and much ingenuity is exercised to see that the coal is loaded carefully, handled as gently as possible and in every handling dropped as short a distance as possible. The whole question of coal breakage is so important that the general term "degradation" is used to refer to the breakage of coal through handling and treatment.

Coarse Crushing (4 In. to ¾ In.)

The first use of machinery in any ore-dressing or coal-preparation plant comes in the coarse crushing. In ore dressing the accepted breakers are of the jaw or gyratory type, of heavy construction to withstand the breaking stresses of the hard ore. In general they have a small ratio of reduction, say 4 or 5 to 1. Many years ago Thomas Edison introduced huge rolls to crush large ore, but the strains and wear on this type of machinery for ore above 2 or 3 in. in size proved so great that the practice was not followed. In coal preparation, where it is necessary to crush large lumps of coal to unlock the contained impurities or to meet market demands for a sized coal, the gyratory and jaw breakers are not in favor, for the following reasons:

1. These breakers have a comparatively small movement, and the coal is so soft that it deforms or allows itself to be dented to the extent of the movement of the breakers without in itself being broken.
2. Many coals and shales, especially if wet, are not gripped by the movement of these breakers, but tend to slide in the breakers during the crushing movement.
3. Some coals tend to become somewhat sticky during crushing, and therefore hang in the throat of the breakers.
4. The coal, being subject to several crushings in the breaker before discharging, makes more fines than is desirable.
5. The coal and shale tend to break into flat pieces in these breakers. This renders the material unsuitable for subsequent sizing.

These factors have led to the introduction of special machines for coarse crushing of coal as follows:

Coal Rolls.—Coal, being soft, can be crushed by rolls with much less power than ore, and with much less wear on the roll faces. For this reason, coal rolls can be made much wider than ore rolls and have a much greater reduction ratio with one pass. Therefore the coal roll is not simply a machine for making a 4 to 1 reduction of the smaller sizes, as with ore, but by means of corrugated roll faces or the insertion of teeth in the roll faces, which greatly increase the angle of nip, the largest lumps of run of mine coal can be reduced to 2½-in. size or smaller with a single pass through the rolls. While the general principle of these rolls is the same as that of ore rolls, there are a number of interesting differences. The ore roll has a separate drive for each of the two roll faces, whereas the coal roll generally has only one drive, the two roll shafts being geared together. Especially in toothed rolls this is necessary, because the teeth in the opposite rolls usually lap, and a differential speed would cause breakage of the teeth. Some coal rolls are of the so-called rigid type, without springs. If springs are used in geared rolls they admit a small movement only, as the gears must not jump out of mesh. The chief danger from breakage comes from tramp iron; to avoid this, some roll makers use a break pin in the driving pulley and some assume that the belt will come off if the rolls stick. Toothed rolls crack the coal rather than crush it, producing a minimum of fines. Speed is a matter of importance, as low speed and consequent tendency to choke feeding makes a larger percentage of fines, although in anthracite practice they have reduced the former peripheral speed of 1000 ft. per minute to 350 to 500 ft. per minute, or to about the velocity of the coal as it enters the rolls. Fewer fines result. In anthracite practice, the tendency is to make a comparatively small reduction in size on each pass through the rolls. This stage crushing produces a minimum of fines. The secondary rolls are called "pony" rolls. The capacity of coal rolls in comparison with that of ore rolls is large. At an Illinois bituminous mine a set of toothed rolls 4 ft. long by 36-in. diameter, driven by a gear-connected 25-hp. motor, crushes 2300 tons run of mine coal to 2-in. size in 8 hours.

The Jeffrey one-roll crusher has a single roll with small teeth; at about every eighth row, four or five large teeth are inserted. Opposite is an irregular fixed or breaker plate with recesses through which the large roll teeth pass. This machine will reduce run of mine coal to 1½-in. up to 4-in. size at a single pass.

The Bradford coal breaker is a sturdily built trommel screen of large diameter. Longitudinal bars inside lift the coal and allow it to drop and break inside the trommel and to pass through the trommel screen. The harder pieces of impurities and foreign material are discharged as oversize. It is useful in breaking soft coal. The Sauerman breaker consists of two oppositely moving jaws with teeth, working against fixed bars. Of the same type is the English Whitaker's coal breaker. The Norton pick breaker, which might be called a mechanical pickax, has been developed to crack the coal with the production of a minimum of fines. Several companies make rotary coal crushers similar in appearance to a gyratory crusher, except that the heads and concaves are corrugated and rotate, rather than gyrate. They will reduce lump coal to 1-in. size or finer.

Medium and Fine Crushing (¾ In. to ⅛ In.)

In ore dressing, "medium crushing" at some plants may include the crushing of particles somewhat larger than ¾ in. and "fine crushing" may include crushing to maximum sizes somewhat smaller than ⅛ in. The limiting sizes given above are therefore relative only. In ore dressing, when it is desired to begin saving the concentrates in as large sizes as possible and to avoid making fines, the smooth-face narrow rolls are still the standard machine for crushing within this range of sizes. The introduction and general adoption of flotation treatment of sands and slimes has decreased the relative importance of rolls during the past few years. First came a tendency to take the coarse rock-crusher product, pass it immediately to ball or tube mills and grind the entire feed to sand and smaller sizes at one operation. Within the past few years there has been a strong reversal to stage crushing. This is partly due to the success of the Symons cone crusher, which has found wide use as an intermediate crusher, taking material from the coarse crushers at 3, 4 or 5 in., or even larger, and reducing it to 1 in. or less. It has a marked advantage of being able to give a reduction ratio as high as 10 to 1. Today, then, standard stage crushing of ores in large plants may involve the following steps:

1. One or two stages of coarse crushing reducing the ore to around 5 in., or possibly less.

2. Intermediate crushing in a Symons cone reducing the ore to 1 in. or less.

3. Rolls in closed circuit with screens, usually of the vibrating type to reduce the ore to ¼ in. or less.

4. Final fine grinding in rod mills or ball mills, often in two steps. This means that the ore rolls are again finding their place in ore-crushing practice.

Stamp mills, the use of which was standard practice in gold milling a few years ago, have been giving way to the ball mill. In any case, we no longer think of the stamp mill as an efficient machine for taking the rock-crusher discharge and reducing it to fine sands and slimes with one operation.

With coal, the field of crushing between ¾ in. and ⅛ in. is dominated by the corrugated roll and the swing-hammer pulverizer, by reason of their large capacity and small production of fines. The corrugated roll needs no description other than that it has a face often 3 or 4 ft. wide, giving it a large capacity. Some corrugated coal rolls are as large as 54 by 54 in. Wear of roll faces is not an important factor. If the corrugated roll is intended for crushing, say above ½ in. maximum, and the roll faces do not touch, they often have independent belt drives and have springs, as ore rolls do; ⅜ in. maximum is about as fine as corrugated rolls will crush. The swing-hammer pulverizer, with its rapidly rotating loose arms, strikes a blow on the coal in space and the broken coal falls away rapidly through a screen without the production of excessive fines. Recently there has been developed a ring hammer mill which does the necessary medium and fine crushing of coal with the production of a minimum quantity of extreme fines. In ore dressing certain soft rock, such as limestone or gypsum, may be crushed with a swing-hammer pulverizer.

Above all, in comparing medium and fine crushing of ores and coal, it should be remembered that in the majority of cases all crushing and grinding of ore after the rock breakers is wet crushing in water, whereas practically all crushing and grinding of coal is dry. Wet crushing aids in ore dressing, not only in movement of material and prevention of injurious dust, but in preparing the material for subsequent treatment. In coal preparation, on the other hand, wetting the coal only adds useless water to the product and often necessitates drying. Most important of all, it is difficult to remove water from the fine coal; altogether water increases the weight of coal without adding to its combustion value. Therefore water is added only to sizes of coal that are to be cleaned by some wet process.

Fine Grinding or Pulverization (⅛ In. to 0 In.)

In ore dressing, it is a natural step from medium crushing to fine grinding, or the production of material most of which passes through

100 mesh. Thirty years ago, every kind of ore-dressing plant avoided the production of these fine sizes or slimes. Today most gold and silver ores ultimately reach the stage of all sliming, and in other ore-dressing plants increased percentages of the ore are being slimed for flotation treatment, even though previous treatment has removed some of the values. Ball mills, rod mills and tube mills of various makes are the standard pulverizing machines. The writer has investigated a number of ore-grinding mills based on other principles than the tube mill. No matter how promising the machine looked in principle, the wear and tear of grinding hard ore made replacement of wearing parts frequent and costly. Only those with the tube mill principle have succeeded for efficient and low-cost grinding of hard ore.

Coal is pulverized or powdered, not usually for treatment but to produce a fuel for cement kilns, reverberatory furnaces and other places where a long flame is necessary. Recently the use of powdered coal as a boiler fuel has been increasing rapidly. The coal is pulverized dry and if the feed has more than 3.5 per cent moisture a mechanical drier may be a prerequisite. The common machines for pulverizing coal are of three types, as follows:

1. The cyclone pulverizer type where inclined fan blades in a rotary pulverizer finely comminute the material before discharging it. The fans create an air current which carries the crushed material out of the pulverizer and through a screen, automatically returning the oversize.

2. The Raymond is a dry-coal grinding mill running on the same principle as the Huntington ore-grinding mill, except that, by suction, currents of air are drawn through tangential openings below the roller rings and are drawn up through the machine, carrying the finely pulverized material with them. Separation of oversize and undersize takes place outside in an air cone, the oversize being returned automatically to the mill. A similar mill in which large revolving steel balls do the grinding is the Lehigh Fuller.

3. The ordinary tube mill run dry is also in common use for pulverizing coal.

All these machines easily furnish a product of which 95 per cent may pass a 100-mesh screen and 85 per cent a 200-mesh screen.

In fine-grinding coal, it is interesting that a coal that is soft and friable in the coarser sizes may be more difficult to grind in the pulverized sizes than a coal seemingly harder in the coarse sizes.

Sizing and Classification

Sizing is an important operation in every ore-dressing, coal-preparation or other mineral-preparation plant. In ore dressing the ore must be screened or otherwise classified in a series of sizes during or after crushing and before it is treated to separate valuable mineral from the waste or gangue. Moreover, no crushing machine will produce a finished product cheaply and continuously unless screens are used. In coal preparation, market requirements, convenience, custom and efficiency in combustion necessitate grading mined coal by sizes, even though no separation of coal from contained refuse is attempted later.

Hoover's translation of Agricola (1556) describes the application of wooden screens, hand-operated, and used either wet or dry, in the ore-dressing practice of the day. Bar screens were introduced at English coal mines about 1740. The four standard types of screens, (1) stationary bar, (2) revolving, (3) shaking, and (4) vibrating, were all introduced first in ore-dressing practice and later adopted and modified by the coal industry. The revolving and shaking screens were at first hand-operated, and later the general introduction of steam engines made possible larger power-operated screens. In 1844 the first power-driven coal screens in the United States were introduced in the Pennsylvania anthracite region. They were "circular" revolving screens with perforated cast-iron plates. Before that time coal was broken by hand on flat perforated cast-iron plates until it was small enough to drop through. In the bituminous coal fields the "Akron" bar screen has been in common use for at least 50 years. The revolving screen formerly used in both anthracite and bituminous practice has given way to some form of shaking or vibrating screen because it caused excessive breakage of the coal. Lately revolving screens have been revived to screen refuse, where the finer particles passing through the screen are of higher coal content than the oversize discharge.

If I were to name any process where coal preparation has gone ahead of ore dressing, it would be in screening practice. Especially since 1900, at nearly every mine the coal shipped to market has been screened. This is often the only or final preparation. Care is necessary to prevent breakage and the tonnage handled is often large. For these reasons many ingenious modifications of the original simple screens are in use and the sizing of coal has reached a high stage of development. Let us compare the screens used.

1. *Stationary or Gravity Screens.*—In ore dressing the stationary or gravity screen is called a grizzly. It is used only to effect a rough sizing of the large sizes, either before or after the rock breakers. It is strong, but does not produce exact sizes. In coal preparation the gravity screen, called a bar screen, which has bars spaced 1¼ in. apart, was for many years the common way to separate lump bituminous coal from the slack or fines. During the past 20 years it has been largely replaced by more exact sizing devices.

2. *Revolving Screens.*—The revolving, or roller screen, called a "trommel" in ore dressing, gives an exact-sized ore product when used either wet or dry. By putting two or more concentric screens on the same shafts, two or more exactly sized products are obtained. For ore dressing

these screens are built heavy and strong, and are more and more replacing "grizzlies" for coarse screening, because of the exact sizes obtained. Where a trommel separates out only one coarse oversize before or after the crusher it is called a "scalper" screen. The increasing need for washing and sizing gravel and crushed rock for concrete making has caused interesting developments in the revolving screen.

Formerly the trommel screen was much used in ore dressing to size material for subsequent jig and table treatment, and is still good practice where jigs and tables have not been eliminated; but the general introduction of flotation treatment has done away with the use of trommels as grading apparatus preceding treatment. The development of the classifier, in efficiency, in capacity and in increasing size range, has also lessened the importance of the trommel. Trommels are still used as an adjunct to the medium-grinding machines.

Prior to 1910, revolving screens were common in coal-screening plants. Since then the development of various shaking screens has made efficient screening possible with less breakage and the revolving screen has nearly disappeared from coal preparation.

3. *Shaking Screens, Riddles or Jiggers.*—In shaking screens, or shakers, riddles or jiggers, a flat screening surface oscillates, causing the material being screened to move over it and find new chances of passing through, or, if too large to pass, for being discharged at one end as oversize. In this class of screens numerous different mechanical movements have been introduced to vary the paths of travel of the coal on the screen surface. In addition to simple shaking screens, which usually are moved with a simple harmonic motion by an eccentric or a crankshaft, and which must be inclined toward the discharge end, this class includes springboard, gyrating, pendulum and differential motion screens, and a number of other types, some of which are horizontal. These screens are large enough to handle up to 1000 tons of run of mine coal per hour, producing several sizes, and virtually the same types are designed for a smaller tonnage of the sizes under 1 in. In the Anthracite Region the large shakers are called "Bull" shakers. It is possible that some of the improved shaking coal screens might find a place in ore dressing.

4. *Vibrating Screens.*—The vibrating or pulsating screen, an offshoot of the shaking screen, has developed into a distinct type. It has an inclined wire-cloth screening surface which is given a rapid vibrating movement of small amplitude at right angles to the plane of the screen surface. This group includes electrically vibrated screens and other types that are given a pulsating movement through cams, ratchet wheels or eccentrics. On account of their rapid vibration they are especially useful in dry-screening the small sizes of ores and coal where the ordinary screen would tend to clog. Lately special punched-plate screening surfaces have been used successfully on vibrating screens.

SCREENING SURFACES

Any of the screens described may have as a screening surface: (1) rods or bars; (2) punched or drilled metal plate or (3) wire screens or wire screen cloth. Rods and bars give an uneven product. Punched or drilled metal plate with round holes is used almost exclusively in coal preparation, as wire screens present an uneven surface and are said to slow up the travel of the coal and promote breakage. In ore dressing wire screens are customary because they give a maximum amount of screening surface and are easily cleaned. Lately, the punched plate is being used more in the heavy trommels.

CLASSIFICATION

Classification of ore in water by free or by hindered settling has long been standard in ore dressing for separation of the finer sizes. It finds little use in coal preparation, where the fine sizes are not so important, although the Dorr drag classifier is used to prepare fine coal for table treatment. However, the principle of classification is used as a final machine for separating coal from refuse. This will be described under that heading.

PROCESSES AND MACHINERY

Having compared breaking and sizing equipment as between ore and coal, we are ready for the processes and machinery used to separate the valuable mineral from the waste. This separation may take place in the larger sizes, in the medium sizes and in the small sizes or fines.

1. *Separation in the Larger Sizes*

Hand picking or sorting is still an accepted method of removing large lumps of waste from the ore before or after the first crushing. The general tendency during the past few years has been to do away with hand sorting, crush the whole product and depend on the concentrating machines to do the work. A notable exception is at the Alaska Juneau gold mill, where a considerable percentage of the waste is hand-sorted before final crushing. The sharpness in distinction of ore and waste as to included particles and freedom from mud and fines; the cost of labor and the relative percentages of clean ore and clean waste in the coarse sizes govern the desirability of hand sorting. It is evident that with labor at 50 cents a day hand sorting would pay where it would not be feasible with $5 per day labor. The writer remembers seeing at Guanajuato, Mexico, women sorting ore of about 1½ in. size as it passed on a belt. Surprisingly few accurate data are available concerning the economy of hand sorting of ores.

In coal preparation, if the coal is not to be washed, an attempt is made to remove the visible shale and other impurities: (*a*) in the tipple during screening, (*b*) on picking bands or belts after screening, and (*c*) during the loading of the railroad cars underneath the tipple. It is generally accepted that the coal must be sized before efficient hand picking can take place; that there must be an inspector to oversee the work, and that the coal must be fairly dry. It is not so much that a man picks out so many tons per day, but often a small number of pieces of refuse spoil the looks of a car of coal and lower its subsequent sale price. Often the amount of refuse removed is under 1 per cent of the product. A hand picker will remove about two tons per hour of refuse when hand-picking coal above 4 in. in size under favorable conditions. When picking 2 to 4-in. coal the refuse removed per man is generally ½ to ¾ ton per hour.

In considering hand picking with both ore and coal, the importance of uniform and adequate lighting must be stressed. It is probable that usually all-artificial lighting is more effective than part natural and part artificial lighting.

Jigs are in use that treat bituminous coal up to 3 or 4 in. in size. With larger lumps hand picking is used where necessary. The egg sizes, say from 2 to 4 in., are hand-picked if the percentage of impurity is small, but are washed if more than 2 or 3 per cent of impurities is present. The nut sizes from ¾ in. up to 2 in. are sometimes hand-picked, although other treatment is more common if many impurities are present. In the anthracite field extensive hand picking, or the use of the "breaker boy," has gone out of existence, except in story books. There are still a few men employed to look over the clean coal as it is loaded and pick out objectional looking pieces which have stayed with the coal in the cleaning machine by reason of their low specific gravity. In the same way men may look over the coarser refuse for pieces of good coal accidentally included. In all coal-cleaning plants that cater to a particular trade, there may be pieces that will pass the washing machines but will not pass visual inspection and which must be removed by hand. In both the anthracite and bituminous fields jigs have been used on coal as large as 6 in. in size. The Chance process also will clean 6-in. coal mechanically.

THEORY OF SEPARATION IN WATER

The theory of separation of heavy from light minerals in water, involving free and hindered settling, is well understood in both ore dressing and coal preparation practice. It follows that sizing or classification should take place before treatment of mixed valuable mineral and waste, if clean products are to be obtained. In ore dressing there have been some interesting departures from the practice of sizing before jigging, as in the Woodbury system and in the multiple cell jig formerly much used in the Middle West zinc region. In these practices the feed was not closely sized, but certain cells in the jig were operated, not only to separate the valuable mineral but as classifiers preparatory to re-treatment of the overflow.

In coal washing, until 1900 it was common to size first and then to wash the coal in the different sizes separately. This practice became known as the Luhrig system. Then developed what is often called the Baum system, where the coal is first washed and then sized and if necessary washed again. Suppose that in the Baum system, after washing, there are small pieces of shale, say of ⅛-in. size maximum, with the clean coal. Screening over a ⅛-in. screen will remove this shale and the small coal and allow perhaps 85 per cent of the product to be shipped as clean coal without other treatment. If necessary, the fines may be re-treated. Thus the Baum system treats together a range of sizes exceeding the theoretical settling ratios of heavy and light minerals. Again, many coal plants, by washing once without sizing, can remove the bulk of the visible impurities and probably meet customer specifications.

The theory of stratification of light and heavy minerals when flowing in water is not yet fully investigated, although several separating processes in both ore and coal dressing make use of the principle of differential stratification.

JIGGING

For many years jigging was the standard and main method of ore dressing. The Harz jig as developed in Germany and introduced to this country was ordinarily used to treat ore up to about 1½-in. size and down to, say, 1/16 in., of course using different jigs in different sizes. Differential-motion jigs were the great improvement made in this country. For the past 15 years, on account of the development of successful treatment processes for fines, the jig often has been abandoned.

Coal jigs were first used in this country in the Anthracite district shortly after the Civil War. Coal, being considerably lighter in weight than ore and gangue, can correspondingly be jigged in larger sizes. Coal jigs have been in operation handling up to 6-in. sizes. For a time, about 25 years ago, much ingenuity was used in developing various plunger and basket jigs and installing automatic draw-offs or waste-removal gates. Usually the refuse or sink product is greater in amount than is the corresponding concentrate in ore dressing, and the coal overflow must be drained and saved. In general the coal jig is larger and of much greater capacity than the ore jig.

The Harz jig for ore dressing is paralleled by the Luhrig, Foust and Elmore jigs in bituminous coal washing and by the Reading, Lehigh Valley, Elmore and Tench jigs in anthracite preparation. They are simple plunger jigs.

The Hancock jig in ore dressing is roughly paralleled by the Stewart, American and Shannon jigs in bituminous preparation and by the Stroh, Christ and Wilmot jigs in the anthracite field. They are simple basket jigs. In ore dressing these jigs are sometimes called movable-sieve jigs, and in coal preparation they are sometimes called pan jigs.

The differential-motion plunger-type New Century ore jig is paralleled by a coal jig of the same name and other jigs, as the James, having patented plunger acceleration movements. The Pittsburgh is a pan type of coal jig with differential motion of the pan. The Baum jig employs air pulsations to give water in the jig a quick up movement. It is possible that the principle of the Baum jig could be employed with advantage in jigging ores.

An interesting development was the Luhrig feldspar jig for fine coal. Here an artificial jig bed of feldspar, having a specific gravity heavier than coal and lighter than shale, was introduced, allowing cleaner separation on the fine sizes. Similar devices were used in ore dressing when jigs were treating fine sizes. With the general abandonment of jigs in the fine sizes, these devices have gone out of use.

Shortly after the Great War, a coal-washing device called the Rheolaveur process was developed in Belgium, which was introduced into this country about 1924. It is now cleaning a considerable tonnage in both the anthracite and bituminous fields here. It replaces jigs. In construction it resembles a launder classifier, but used as a medium for final separation and not classification. The impure, sized coal flows in water down a narrow trough. The heavy flat slate soon is sliding on the bottom. At the appropriate place the bottom of the trough is cut by the insertion of a classifier pocket with upward rising water. The slate and refuse drop into this pocket and are removed. The lighter coal passes on. By proper design it is possible also to insert a pocket that will remove middlings or bone coal, which may be retreated. This process has been successful not only in replacing jigs but in greatly increasing the capacity of a plant per unit of space. The principle of the process involves the stratification ratios of the materials treated as well as the settling ratios. More research work is needed on the subject of stratification ratios. There seems to be no reason why the process should not be successful with certain classes of ores as a means of replacing jigs and other coarse separating machines.

In the development of coal washing, the Robinson tub washer was used extensively 20 or more years ago, although today it has been replaced by more exact cleaning devices. It consisted of a large cone filled with water, and agitated by stirrer arms. Following the classifier principle, water was introduced at the bottom; dirty coal was fed in at one side on the top and traveled across the agitated mass. Heavy refuse fell to the bottom and was removed periodically through a double valve, while the clean coal discharged with the excess water at the top at a point opposite the feed. It was customary to feed the washer with unsized coal, with the result that considerable fine shale overflowed with the large coal; or, conversely, some clean coarse coal fell and was discharged with the refuse. There is no similar machine for ore dressing, although the Richards-Janney classifier uses somewhat the same principle.

A distinct American improvement in ore dressing was the introduction in the late nineties of the concentrating table for treating sizes under ¼ in. and larger than 60 mesh. Different tables have different shapes and different arrangement of riffles. Ingenious head motions, as illustrated by the Wilfley and Deister, give the tables a slow forward motion and a quick return, which results in moving the material towards the tail end of the table. By 1910, concentrating tables had become common in nearly all types of ore-dressing plants. Probably the first attempts to use the concentrating table in a coal washery were at Dawson, N. Mex., in 1906–07. In 1913, twenty-four Wilfley tables were placed in operation at this plant. In 1914 the writer conducted experiments in cleaning coal by use of the concentrating table. The results were so favorable that shortly afterwards the Deister table was redesigned and placed in successful operation in cleaning the small sizes of anthracite. For coal work it is necessary to have higher riffles, because sizes up to ½ in. are amenable to table treatment. It was also necessary to greatly increase the capacity of the table. In general tables have been successful in anthracite preparation, making possible the treatment of sizes not well handled in jigs and building up the market for fine, clean coal. In many bituminous coal washeries, the larger percentage of the coal is cleaned in the coarse sizes, and many operators have felt that the table did not justify its expense in view of the small percentage of the coal suitable for the table. However, in a considerable number of modern plants, especially in those of large capacity, the concentrating table is in use.

Mention should be made here of the treatment of middlings. It is well known that any ore-treatment process on coarse or fine ores produces a middlings product, which is usually recrushed and re-treated. Likewise in coal preparation any treatment process produces a middling known generally as secondary coal, often as bone coal, which is a product intermediary between good coal and clean refuse. Sometimes this product may be crushed and again treated as in ore dressing; often the mixture of coal and shale is so intimate that recrushing does little to unlock the contained coal. Thus, secondary coal is often burned locally as a low-grade fuel. Altogether, in coal preparation, cleaning is more relative than absolute. A coal containing, say, 12 per cent ash and 2 per cent sulfur, may be washed to produce a refuse of 60 per cent ash and 6 per cent sulfur; a middlings or secondary coal of 25 per cent ash and 3 per cent sulfur; and a clean coal of 7 per cent ash and 1 per cent sulfur. Any attempt to get cleaner coal results in throwing more coal into the secondary coal and refuse, and vice versa.

CONCENTRATION OF FINE SIZES AND SLIMES

In ore dressing, the concentration of fine sizes and slimes by flotation has become almost universal. Whereas those who practiced ore dressing 25 years ago were using every possible method to avoid the production of slimes, today it is common to crush all the ore to sizes that can be treated by flotation. Each year of the past 25 has seen progress not only in the development of flotation apparatus but in the introduction of new reagents and technique, which have resulted in the application of flotation to ores formerly too complex or too low grade to justify treatment. With coal, flotation treatment has been slower in development, largely because of the low value of clean coal fines. Added factors have been the cost of removing water from the fine coal and the small percentage of the coal in a treatment plant that would be in sizes amenable to flotation treatment.

Perhaps the first scientific experiments on coal flotation in this country were made at the Mellon Institute of Industrial Research in 1917 and 1918, of which the results were published in 1919. They showed that coal flotation was technically feasible. In the past 5 or 6 years several coal-flotation plants have been operated in England, a considerable number in other foreign countries, and in this country the Pittsburgh Coal Co. has operated a small plant successfully. There is no doubt that, technically, fine grinding of many coals, with the attendant unlocking of impurities, can result in a cleaner coal product. However, the question of when and how this product may compete in the market with a less clean coarser coal is quite another problem. At the present time, approximately 100,000 tons of coal per year are treated by oil flotation in England, and a total of two millions of tons annually in all countries. Its use will be extended where the clean coal must have a guaranteed ash that is practically the same as the inherent ash of the coal.

In the anthracite field the hard, very fine sizes are being treated successfully by the Hydrotator process, which employs certain of the principles used in counter-current decantation.

SPECIAL APPARATUS

Ore dressing has its own special apparatus, which can have no parallel in coal preparation; for example, magnetic separation and amalgamation, although magnetic separation has been used in Germany to separate unburned coal from ashes. In the same way, coal preparation has developed highly a special type of preparation apparatus that will be considered under the name of "Separation by Heavy Solutions."

Separation by Heavy Solutions means the employment of a liquid medium of sufficiently heavy specific gravity so that the coal will float on its surface while the refuse will sink. In ore dressing, if an ore containing wood chips is thrown into water (sp. gr. 1), the wood chips float and the ore sinks. In bituminous coal preparation, if a solution of say 1.35 sp. gr. can be used it is evident that all coal particles of a lower specific gravity will float and that all dirty coal and refuse particles of a higher specific gravity will sink. (Fairly pure bituminous coal has a specific gravity of about 1.30.) Likewise, in anthracite preparation, where the purer coal particles have a specific gravity around 1.5, separation of coal and refuse may be had in a solution of about this specific gravity. It also should be borne in mind that theoretically the separation is independent of size; that is, that a large and a small coal particle will float together. For this reason no sizing should be needed before separation. Practically, there are difficulties in separating small sizes, say under ¼ or ½ in., at the same time that the larger sizes are treated, so that in coal separation by heavy solutions a single preliminary sizing of this character is usually made. To make the same process applicable to ore dressing we would have to find a cheap solution of about 2.70 (the usual specific gravity of gangue). While some experimental work has been done in separating ore from gangue with the liquid acetylene tetrabromide (sp. gr. about 2.9) it cannot be considered a commercial process today.

In coal preparation, zinc chloride added to water produces a solution anywhere from 1.00 to 1.8 sp. gr. Actually this solution is too costly to be used for commercial separation of coal from refuse, but it is used in coal laboratories not only to find the theoretically perfect separation possible at any specific gravity, but as a quick and cheap method of checking washery performance. A solution of carbon tetrachloride is also used for the same purpose. It is possible that in certain ore-dressing processes, solutions as of bromoform or acetylene tetrabromide could be used for accurate and rapid analysis of the work of the concentration machinery.

For practical coal preparation the standard wet method of using the equivalent of a heavy solution is the Chance process. Here a mixture of sand and water is kept in suspension, and this mixture acts exactly like a heavy liquid, having a specific gravity equal to the density of the solid-water mixture. In operation the process uses a large cone in which is agitated a mixture of sand and water. The apparent specific gravity of this mixture may be up to 1.7, depending on the amount of sand in the mixture. Raw coal is fed into the top of this cone, and the clean coal quickly floats over the opposite edge, while the heavier refuse sinks and is drawn out at the bottom. The sand and water overflowing with the clean coal is screened out and returned to the cone. The apparatus has been widely adopted in the Anthracite region, because of its large capacity, less breakage and lessened mill space as compared with screens and jigs. The process requires a preliminary screening at about ⅜ in., for if this is not done there is a tendency for fine coal to be dragged down into the refuse and for fine refuse to adhere to the coarse coal. The process has made some headway in the treatment of the harder of the bituminous coals. The Wuensch process is similar in principle to the Chance, but circulates a mixture of water, fine coal and waste sludge. In the Middle West zinc district, separation of zinc mineral from gangue has been obtained by circulating a mixture of water and slimes obtained from the crushed ore.

If an air blast is allowed to bubble up through a bed of sand, the whole mass boils and dances and has an apparent specific gravity up to 1.5. The air-sand process uses a sand bed of this character to clean bituminous coal. Its advantage is that it treats the coal without wetting it. In attempting to adapt these processes to ore dressing, it must be remembered that, since ores are heavier than coal, heavier solutions or their equivalents in a solid-liquid mixture would be needed. So far, this factor has limited their extended practical application in ore dressing.

Air Tables

Some 30 years ago placer miners, to separate gold from contained sand in desert regions where there was no water, tried using an air blast to blow away the sand. Out of this there developed a dry concentrating table for ores, using air as the separating medium. While the great difference in specific gravity between sand and gold allowed its application to certain conditions of placer mining, it did not find general application in ore dressing, owing to the necessity for close sizing where mineral and gangue differed little in specific gravity and because of the dust produced. For example, if ore had to be crushed to 1-mm. size for dry table treatment, a considerable percentage would be smaller than 100 mesh, which the table would not treat satisfactorily. Some 15 years ago the principle was applied to coal preparation, and with various modifications has continued to find favor, largely because it treats the finer coals without wetting them. Because of the explosibility of coal dust, rather elaborate dust-collection systems are a necessary adjunct to the process. Later tables, by allowing a bed of fine coal to remain on the bottom of the table and blowing the air through this bed, have gained something of the heavy-solution effect of this air-coal mixture and correspondingly decreased the necessity for very close sizing. Perhaps also the market for the product of the average coal-cleaning plant does not require such purity of clean valuable mineral and waste as does the ore-dressing plant. Thus a range of sizes exceeding the theoretical range may be treated together and yet the clean coal as a whole may pass market demands. Again, owing to the heavy specific gravity of ores, it is doubtful whether these processes can be applied generally to ore dressing.

SEPARATION BY DIFFERENCE IN FRICTION

The writer well remembers his surprise, many years ago, when cleaning out a coarse-ore bin through which many thousands of tons of gold ore had passed, to find the accumulated fines at the bottom very rich. There had been a gradual concentration of the small heavy particles at the bottom of the bed, and the larger pieces had for years slid over these fines, without dislodging them. It is only a step from this to imagine an ore-dressing process in which, by reason of difference in sliding friction between two different minerals, the one that slides most easily on an inclined surface should be separated from the one with a larger coefficient of friction. In ore dressing this frictional separation seems not to have been successful, but in coal preparation this principle has been applied with success in anthracite preparation and in several cases in bituminous coal preparation. The Pardee spiral separator is the most widely used apparatus of this class. It consists of several concentric sheet-steel spirals designed with an angle great enough so that the flat heavy slate will travel slowly downwards. The coal, being lighter and with less frictional resistance, accelerates over the same path and is thrown into an outer spiral by centrifugal force. Where the coal is hard and the slate is flat and not in excessive amounts a satisfactory separation may be made providing a sized feed is used. It has the added advantage of treating the coal dry.

The Herrisford process, as developed in England, operates by bouncing the coal and slate on glass and effects a separation due to the differential bouncing characteristics of the minerals.

There are possibly other methods of ore dressing, based on some physical property of the materials, which might be applied to coal preparation. For example, the Huff electrostatic machine will make a good separation of coal from slate, but it does not look commercially attractive for this work on account of cost and capacity limitations. The writer does not know of any other method of coal preparation besides those already described.

AUXILIARY APPARATUS

Conveyors and Elevators.—In both coal and ore dressing the early chutes, conveyors and elevators were generally constructed of steel. At least 30 years ago the ore-dressing industry found the wear of the ore on the steel excessive, and rapidly adopted rubber belt conveyors and elevators. The coal industry was curiously slow in following this practice, possibly because the wear of steel by coal was less and the movable steel conveyors and elevators seemed to be doing satisfactory work. Within the past 20 years, however, the coal industry has rapidly adopted rubber belting for conveying and elevating machinery, with generally greater life and satisfaction. Even today, however, it is interesting to note that the average sand tailings pile is fed from the mill by a series of rubber conveyor belts, while a refuse pile at a coal-preparation plant is fed generally by a refuse car running up an inclined track and dumping on top of the pile.

Prevention of Breakage.—Coals vary greatly in friability. In coal preparation, the breakage of coal in handling is given the general term "degradation." The importance of preventing breakage already has been mentioned. In designing coal-treatment plants, it is most important to handle and drop the coal as little as is possible. Chutes are designed with a minimum of drop. Storage bins are provided with step chutes, which allow the coal to drop gently into a partly empty bin by a series of steps instead of a single drop. Spiral chutes are used where otherwise the drop would be damaging. Often chutes of brass or other metal are used to get a coefficient of friction that will move the coal without accelerating it. Inclined traveling steel conveyors, called "loading booms," are used to lower coal into the railroad car. Inclined shaking chutes often lower the coal gently whenever it goes to the next step in the cleaning process. I know of no corresponding care needed in handling ores.

Dewatering Equipment.—Thirty years ago the average ore-dressing plant sent its water and slimes into the nearest stream, or if it was desired to re-use the water, a crude log dam was built to impound the tailings and return to the mill the water that seeped through. At the same period coal washeries were sending into our streams great quantities of "black water" charged with fine coal. Even today many tons of fine coal are recovered each year by dredging the old sand bars in the streams below the anthracite washeries. Then the ore people introduced large classifiers with the purpose of drawing off a fairly clear water overflow and a thickened pulp underflow. In coal preparation an independent development of large settling cones took place, as for example, the Ramsey sludge tank. During this period, there was increasing friction between the mills and the people having interests in the stream below. In a large way, the solution of the problem came with the introduction, about 1908, of the Dorr thickener in ore-dressing practice, which in many cases allowed a continuous overflow of clear water and a continuous discharge at the bottom of thickened slime. This thickener has been adopted in other lines of industry, and during the past 15 years has been increasingly used in the coal washeries. In ore dressing there are cases where the water is still colored after settlement, even though the bulk of solids has been removed. So in coal preparation the water discharge is often black even though most of the solids have been removed. Interesting experiments have been under way during the past few years in Pennsylvania, endeavoring to clear up this so-called "black water." Today the more modern washeries in western Pennsylvania do not discharge any of it. All the sludge is settled in Dorr thickeners and the water overflow is recirculated in the washery. The time is near at hand when water used in all ore-dressing and coal-dressing plants must be returned to the stream in a clean condition. There should be progress here during the next few years in both ore and coal industries.

Driers.—One of the writer's early lessons in practical ore dressing was the fact that while ores could be crushed and treated either wet or dry, yet there was an intermediate state of stickiness where the fines adhered to crusher, belts, conveyors and rolls. If the ore is to be crushed and treated wet, this is soon overcome. If, however, the plant is designed for a dry treatment, some sort of drier must be introduced early in the process. In ore dressing a cylindrical drier is easily installed and used. Again, in a few cases, fine concentrates are dried in a similar way before shipment. In crushing damp coal, there is less trouble. The coal is generally in larger pieces as compared with ore, and the surface water is often absorbed as new faces are presented during crushing. It is at the end of the treatment that water in coal gives the most trouble. As a comparison, I remember handling arsenical pyrite concentrates of maximum size of 26 mesh. Heaped up on a floor they would drain to 8 or 10 per cent moisture during a single night. Bituminous coal of this size would form almost an emulsion with the water and refuse to drain.

Fine sizes of concentrates are successfully dried for shipment by dewatering with a vacuum filter. The development of the several types of filters for dewatering fine material was brought about largely by the need of cyanide plants for removing solutions from slimes.

In the past three years, for dewatering very fine coal, Oliver, Dorr, America and Genter filters have been installed at several bituminous-coal washing plants in western Pennsylvania. In several cases fine coals are dried by mixing them with some coarser coal in a centrifugal drier, of which there are at least two makes on the market. A recent interesting development is the adaptation of the Dwight-Lloyd sintering machine for drying fine wet coal. It is also possible that we will be able to reduce the inherent high moisture content of certain coals by use of this machine.

Fine coal is dried occasionally in a revolving drier externally heated, or by steam tables. In any case drying fine coal under ¼ in. in size is a troublesome operation. For the sizes larger than this, drain bins or drag screens are used after washing to remove the excess moisture from the coal. It should also be remembered that an open railroad car loaded with wet sized coal will drain itself before reaching its destination. The "wettability" of different coals of different sizes is an interesting study.

PLANT DESIGN

The design and erection of ore-dressing plants generally are entrusted to specialists, employed directly by the mining company for which the plant is being built. Machinery and equipment may be purchased from many different manufacturers. This method is logical because of the great difference in the character of ores. One can almost say that no two mines should have identical plants if the best recovery of values is the requisite. Certain manufacturers of ore-dressing machinery do design and install complete plants, but generally at the smaller mines, of which the owners cannot afford to undertake the research and design cost necessary for an individual plant.

In coal preparation, on the other hand, past practice has left the design and erection to a machinery-manufacturing company or to a company specializing in complete coal-preparation plants. We might say that ore dressing buys tailor-made suits, while coal preparation uses ready-made suits. In both methods there is merit. Undoubtedly much more preliminary research has been done before the average ore-dressing plant is designed than for coal plants. Many coal plants are equipped with certain machinery simply because the manufacturing company that obtained the contract happened to be the builders of that machinery. At the same time it should be remembered that many small coal companies are not financially able to employ independent coal-preparation specialists directly and the manufacturing companies have been able to provide them with standardized plants of real merit at a low cost. An exception to this rule is found in the great anthracite breakers, which are generally company designed, and in the newer very large bituminous treatment plants.

Large ore-dressing plants generally are designed in several independent units or sections, so that if operated at part capacity some units may be closed down while the others operate at full capacity. In coal preparation, a plant of equal capacity generally is a single unit. If it operates at part capacity, serious inefficiencies may arise. It must be remembered, however, that the capacity of individual coal-crushing and treatment machinery may be 600 tons per hour, making it difficult to sectionalize even a large plant.

Ore-dressing plants generally are designed with coarse-ore storage bins, and operate continuously on a 24-hr. day. Coal-preparation plants take their feed direct from the mine cars and generally operate an 8-hr. day only. A storage bin would cause serious breakage. Where a coal-preparation plant serves a number of mines and the raw coal is hauled to the plant in railroad cars, it is possible to use these cars for temporary storage and to extend the operating time per day. The difficulty of hauling mine cars for a long distance on the surface, or the loss from the added breakage that comes from dumping into and out of railroad cars, has caused the erection of most coal-preparation plants at or near the mine mouth, without reference to the excellence of the site, while ore-dressing plant designers consider the site of prime importance. It is possible that economies can be brought about in the construction and operation of coal-treatment plants by giving more attention to plant location.

The product of an ore-dressing plant enters a fairly standard market. For example, zinc concentrates are generally of a minimum standard grade, with penalties for impurities, although they may come from many different mines. Coal cleaning, on the other hand, may take place at captive or at commercial mines. By a "captive" mine we mean, for example, a mine owned by a steel company and producing coal only for the needs of that company for coking and other purposes. The product never enters the general market. A large part of the advancement in plant design has come from these captive companies, as they could afford to spend time and money to bring their own products to a high degree of purity.

General commercial mines, catering to a diversified trade, have only recently been forced to meet rigid specifications by the larger general consumers. Thus the design and operation of coal-preparation plants is closely allied to the selling or marketing end of the coal business, while ore-dressing plant design leans more to the technological end.

Conclusion

Ore dressing or milling and coal preparation are based on the same principles and many of the machines used have, with modification in design, been used in both industries. In most cases ore dressing has led in invention and application, but during the past 15 years there has been remarkable technical progress in coal preparation. It should not be forgotten that ores are inorganic chemical substances of fixed chemical composition, while coal is a complex organic substance containing an almost infinite number of chemical compounds. There may be processes and machinery now used in the ore-dressing industry which can be applied with success in the coal-preparation industry, or the reverse may be true.

Acknowledgments

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DISCUSSION

A. ANABLE,* New York, N. Y. (written discussion).—Dean Holbrook's paper is most interesting to those who have had contacts in both coal and ore. Although "coal dressing is ore dressing upside down," there is a common bond from the use of certain similar operations or processes and from the infiltration of metallurgists and their ideas into the coal industry and vice versa.

Classification.—The mechanical classifier plays a minor role on anthracite in dewatering fine coal from tables. It is also occasionally used for dewatering further the discharge from thickeners and hydroseparators where this fine coal is to be cleaned by Rheolaveur or else marketed direct. Its purpose on coal is to produce a coarse rake product as free from moisture and sludge as possible, without much regard for the character of the fine overflow; this is just opposite to the objective on ores. Comparative figures are: for ores when operating at 20 strokes per minute, separating at about 65 mesh, overflowing at close to 4 to 1 dilution, and discharging rake product at 15 to 20 per cent moisture, the tonnage raked is 200 to 300 tons per 24 hr. per foot of rake width; for anthracite under same conditions of operation the overflow dilution is about 10 to 1, the moisture in the discharge 20 to 30 per cent and raking capacity 70 to 90 tons. The moisture content of rake product and the raking capacity are both more favorable on ores because of higher specific gravity (2.65 versus 1.75) and the denser overflow (4 to 1 versus 10 to 1).

Thickening.—The problem is broadly identical in both industries—to deliver a practically clear overflow. Mills usually run 24 hr. and coal plants rarely over 16 hr. per day. Overflow in mills must be practically clear water, while in an anthracite breaker the water may build up to 3 to 5 per cent of minus 200-mesh solids in the course of 16 hr. During the interval of shutdown these solids settle in the thickener and go out with the underflow when the thickener starts again. On ores the settling area ranges from 8 to 12 sq. ft. per ton of solids per 24 hr., feed dilution is 4 or 5 to 1 and moisture content of discharge is 40 to 50 per cent. Corresponding figures for coal are 4 to 6 sq. ft., 20 to 1 dilution and 60 to 65 per cent moisture. The largest units built to date for coal are 150 ft. in diameter, while on ores 200 and 225-ft. diameter are common, and there are two units 325 ft. in diameter. The greater capacity on coal is: (1) because complete clarification is unnecessary; (2) because the particles though lighter are more granular and quicker settling; and (3) because settling occurs in a more dilute pulp. If coal were ground as fine as ore and a clear overflow demanded, it is probable that the relative capacities would be reversed.

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Filtration.—In base-metal mills continuous vacuum filters dewater the concentrates before smelting. In cyanide plants they recover the last of the solution from the tailings. In bituminous practice with Rheolaveurs they sometimes dewater fines minus 20 or 28 mesh for coking. On minus 100-mesh flotation concentrate and minus 200-mesh cyanide tailings, vacuum-filter capacity runs up to 2000 lb. of solids per square foot per day and the filter cake contains 15 to 25 per cent moisture. Corresponding figures on bituminous coal, which is classed as a free filtering material because of relative coarseness and porosity, are 1800 lb. per day and 19 to 25 per cent moisture.

Hydraulic Classification.—It has long been good practice to prepare a classified feed for ore tables. Until comparatively recently, anthracite practice merely classified out the sludges by sending undersize of 3/32-in. round screen to a hydroseparator which sent plus 48-mesh underflow to tables and minus 48-mesh overflow to final culm thickener. Some recent tests with a Fahrenwald six-pocket classifier, feeding three tables, showed 95 per cent of the feed to Table 1 to consist of coal between 1/16 in. and 28 mesh and of refuse between 14 and 65 mesh; 90 per cent of the feed to Table 2 consisted of coal between 20 and 48 mesh and of refuse between 28 and 100 mesh; and 90 per cent of the feed to Table 3 consisted of coal between 28 and 100 mesh and of refuse between 48 and 200 mesh. Technically the test was a great success in that the classified feed increased the over-all recovery of fine coal 30 per cent, increased the table capacity 50 per cent, and gave a coal product with materially less ash and a slate product high in ash, but economically it was a disappointment in that the anticipated market for the improved product failed to develop.

It is often claimed that the removal of water from fine coal is difficult and expensive and this has delayed the exploitation of the finer sizes. The actual facts are: (1) that in thickeners in anthracite preparation 4 to 6 sq. ft. of settling area do the work of 8 to 12 sq. ft. in ore dressing and 60 per cent moisture in discharge on coal is not very different from the 50 per cent on ores; (2) that in filtration the cost for 2000 lb. per sq. ft. of area and a 15 to 25 per cent moisture cake in ore dressing is only slightly different from the cost on coal with 1800 lb. per sq. ft. and 19 to 25 per cent moisture; (3) that 3 to 5 tons of water is used per ton of coal as against 4 to 4½ tons of water per ton of ore. The conclusion is that dewatering is no more difficult on fine coal than on ore but it has become a standard, indispensable, prime operation on ores whereas on coal it is far from standard and is classed as dispensable auxiliary.

The solution of the fine-coal dewatering problem does not so much await the development of new methods as the development of an anthracite market for these fine sizes that will encourage the coal operators to study more closely what is now being done with available apparatus in the field of ore dressing.

R. W. Arms, Congress, Ariz. (written discussion).—An amplification of Dean Holbrook's reference to the Wuensch process may be of interest since this process is now in operation in two large coal plants in Kansas and in a small pilot ore-milling plant.

The underlying principle is a heavy-density solution built up by mixing finely ground particles of the heavier constituents of the coal or ore with water until the mixture will float the lighter particles of the feed. While this principle is not new, the Wuensch process overcomes many of the difficulties met in previous applications of this principle.

Since separation occurs in a suspension medium maintained with a minimum of agitation, only a slight difference in specific gravity between valuable mineral and waste is necessary for separation. It is possible to eliminate large amounts of coarsely crushed waste at the head of the mill. For example, in a large-scale test on Joplin zinc ore, using galena flotation concentrate as a medium, it was found that 80 per cent of the tailings could be eliminated at 1-in. crushing with a tailings loss even less than standard milling practices involving ¼-in. crushing. This was made possible by the rather coarse sulfides in the ore and the uniform specific gravity of the gangue.

In the case of coal this process produces finished coal and finished refuse and is independent of size. Feed from 6 in. or more to zero may be handled in a single operation. On ores its development so far has been confined to coarse materials to remove a finished tailing but not to make a final concentrate. Since this operation is cheap and calls for little capital outlay, it is attractive as a means of increasing mill capacity. On the Joplin ore mentioned above the simple preliminary or roughing treatment by the Wuensch process eliminates 80 per cent of the waste material and leaves only 20 per cent to be further treated in the finishing mill.

W. H. Coghill,* Rolla, Mo. (written discussion†).—The author says that coal is complex. Truly coal is complex and the sulfide ores have a fairly fixed composition, but the ore-dressing engineer must here include the iron ores. They are complex. Many of them contain insoluble matter that is as inherent as is ash in coal. The future iron ores of the United States are so complex that when concentrated to yield a furnace feed with 8 per cent of SiO_2, the tailings contain from 20 to 40 per cent of iron as intimately associated with gangue as is coal with bone. The ratio of product to waste is comparable to that obtained in coal washing. The following statistics show the importance of iron ores in our future ore dressing: In 1929 we required for consumption 73,000,000 tons of iron ore and in the same year we milled 89,000,000 tons of sulfide ores. The latter, of course, are of "fixed composition." Thus, about half of our ores are complex and half are simple. When the field of ore dressing has reached further into the realm of nonmetallics it will have to be remembered that metallic oxides of the sedimentary type are usually complex.

The reversal to stage crushing mentioned by Dr. Holbrook is only incipient. Only expediency can justify the practice indicated by some of the Bureau of Mines milling papers requiring a ball or rod mill to struggle with a 1-in. feed while reducing the fines to flotation size.

As for "stamp mills . . . giving way to ball mills," if a certain news item[1] is correct, the steam stamps of the Michigan copper country will give way to cone crushers. This item reads: "Experimentation has been done on amygdaloid copper rock with cone crushers, and the results are said to be promising. The principle difficulty has been in the malleability of the copper, the large and small pieces merely changing in form while the rock is crushed. By sorting out the pieces beforehand, however, and using a series of cones in closed circuit, a method may be adopted that is more efficient and economical than the present steam stamps."

On page 136, there is the statement: "In fine-grinding coal, it is interesting that a coal that is soft and friable in the coarse sizes may be more difficult to grind in the pulverized sizes than a coal seemingly harder in the coarse sizes." We have been similarly impressed in grindability tests on coal.

Then again, carborundum, which is reputed for its hardness, crushes easily from 8 to 48 mesh but below this it becomes resistant. These examples show that molecular structure must be studied along with grindability. Mineral physics should be given more attention.

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† Published by permission of the Director, U. S. Bureau of Mines.

[1] *Eng. & Min. Jnl.* (1932) **134,** 596.

When he says, page 139, "classification of ore by free or hindered settling has long been standard in ore dressing . . . ," Dr. Holbrook doubtless intended to include hydraulic classification. Probably only a few of us are aware that hydraulic classifiers with from 20 to 25 cells are in operation with tables. The metallurgical results on the tables are far superior to those obtained by the former practice with distributed feed—that is, where the same kind of feed goes to all of the tables. The earlier work was deficient because classification was neglected; but now classification is considered as a link in the surrounding flow sheet. When the first tables take a concentrate from the classified feed and return the coarse gangue to be reground with the new feed, and only the last tables make a tailing, obviously the feed to the last tables is so much impoverished in mineral that remarkably low-grade tailings are a natural result. This should not be overlooked in the treatment of complex ores where the lead contaminates the other sulfide minerals in flotation. Another point of no small importance is that by this flow sheet a remarkably well prepared circulating load to the ball mill is made. One of the classifiers mentioned handles 2000 tons per day of feed which consists of new ore and circulating load.

There is another hand-sorting plant which should be paired with the Alaska Juneau, mentioned on page 139. It is in the Tri-State zinc district. The only essential difference is that at the zinc concentrator the mineral is removed by hand and the barren flint goes to waste from the discharge end of the belt. The material milled represents less than two-thirds of the tonnage hoisted.[2]

On page 142, the author says: "There seems to be no reason why the process (Rheolaveur) should not be successful with certain classes of ores as a means of replacing jigs and other separating machines."

The Rheolaveur was given a trial in competition with rougher jigs in the Tri-State zinc district. It was not successful. On the other hand, the Rheolaveur process has been in operation in the Minnesota iron district for several years. There it concentrates iron ores too fine for the jigs.

In further consideration of this subject, the high-intensity induction magnetic separator will have to be included. The induction machine makes a better separation because the magnetic field is applied to a falling stream instead of being applied to a mass of ore at rest.

G. R. DELAMATER,* Cleveland, Ohio (written discussion).—Dean Holbrook has well stated the status of coal preparation and its relation to ore dressing. Treatment of fine coal is still largely an unsolved problem. If done dry, the collection of fines is expensive but if done wet the cost of dewatering the product is almost prohibitive. The ore dresser has done little in dry concentration to guide the coal man and many wet operations on ores are inapplicable to coal on account of cost.

Milling plants are in general large and costly to construct. Highly specialized talent is employed in their design and construction while coal plants are usually bought from machinery firms on a competitive basis without as much preliminary study and investigation as the problem deserves.

The division of a coal into 12 to 18 sizes between 1 in. and 200 mesh and a study of the characteristics of each size will often show that maximum elimination of impurities does not necessarily mean the greatest profit. It may be found more advantageous to clean certain sizes and bypass other sizes to be mixed with the cleaned products.

Vibrating screens are making notable gains in the field of fine screening of coal around $3/8$ to $3/32$ in., even though the coal has a rather high moisture content. Capacities of feed per square foot of screen area as high as 2 tons per hour at 3⁄32 in. and 5.7 tons at 3⁄8 in., have been attained.

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[2] C. O. Anderson: Recent Practice in the Milling Procedure in the Tri-State District. *Min. Cong. Jnl.* (1929) **15**, 850.

* Assistant Vice President, The W. S. Tyler Company.

While Illinois led all other states in amount of coal washed 20 or 25 years ago, it has practically disappeared in that state today, largely because cleaning did not pay for itself in increasing the sales price of the coal.

Regarding the statement that "today there is no acceptable standard as to what constitutes clean bituminous coal," such standards for general use can be established only as to method of determination. Competitive conditions, general characteristics of the coal, location of the mines and similar matters would alter the standards from any general formula that might be established. After all, would not the evaluation of coal on a heat unit basis together with its adaptability to coking, gas producer, steam, etc., automatically establish a clean coal standard according to use and location?

Swing hammer pulverizers are really crushers and are not suitable to produce pulverized fuel. Their field is more on coal after cleaning, particularly to produce a uniform granular product with minimum extreme dust for byproduct oven use. They do not operate as pulverizers "without the production of excessive fines," at least as far as coal is concerned.

There is no doubt about the increased capacity and efficiency obtained by the use of screens ahead of the crushers or in closed circuit with them.

The setting by Dean Holbrook of 1.30 as the specific gravity of fairly pure bituminous coal seems too low; 1.45 is more nearly representative of average conditions, meaning all coal that will float on a 1.45 sp. gr. solution.

T. Fraser,* Cleveland, Ohio (written discussion).—The statement of Dean Holbrook that the Hydrotator process employs certain of the principles used in countercurrent decantation was true for a few anthracite plants built several years ago but does not apply to the modern Hydrotator washer, which has been evolved from the L. C. Trent agitator and classifier originally designed for ores.

While many coal-washing devices have been adapted from the older art of ore dressing, a stage has been reached where a marked flow of ideas is going in the opposite direction.

Ore dressing had to overcome great technical problems but coal washing had not only equally great technical problems but also economic difficulties which retarded technical progress until recently. A marked difference in scale of operations existed in the past, as evidenced by the rating of ore mills in tons per 24 hr. and coal plants in tons per hour. However, conditions for ore and for coal are rapidly approaching the same status because coal is receiving more technical attention of late and ore dressing has had to be applied to very large tonnages of low-grade ore, approaching the narrow operating margin common in coal-preparation projects. Hence it appears that ore dressing and coal preparation have more in common now than ever before.

R. Gahl,† Berkeley, Calif. (written discussion).—The paper seems to call not so much for a discussion as for an expression of appreciation for such an able comparison of the machinery which coal and ore-dressing engineers have adopted to solve their problems. Often the problems are identical and call for adoption of identical machinery, as was strongly impressed upon me by a personal experience some years ago. I was retained to pass on plans for a coal-washing plant and objected to the use of an enormous settling cone where a Dorr thickener seemed to be the logical apparatus. However, I was unable to convince the manufacturers who had prepared the plans that my suggestions were sound, for they approached my clients and declared to them that if I had anything to do with the design of the new coal washer they would not want to construct it. I hope that such a thing could not happen any more at the present time.

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† Consulting Metallurgist.

B. W. Gandrud,* Tuscaloosa, Ala.—I want to emphasize the tremendous advantage which the coal operator has in the simple float-and-sink method of testing and control, especially as contrasted to the ore dresser, who requires solutions of higher specific gravities, 2.5 to 4.00, which are too expensive to use in the large quantities required by coarse pieces of ore. From our experience with Alabama coals the washability curves obtained from float-and-sink tests enable us to predict washery results so accurately that the ash content of the washed coal will not vary more than 0.5 per cent above or below the estimate from float and sink and the recovery of washed coal will be within 2 or 3 per cent of that predicted. In making these predictions the type of washery must be taken into account, since one type of plant will give somewhat better results on certain sizes of coal than other types. In the operation of a plant the float-and-sink data show whether the plant is operating efficiently. Lack of this simple float-and-sink method for ores means that other means of control must be used and that a pilot plant may be called for to determine the amenability of the ore to concentration and even the pilot plant may not give the same results as are obtained later in regular mill operation.

Dean Holbrook failed to mention the Montgomery jig, which is used in about 60 per cent of the coal-washing plants of Alabama. It is a pulsion jig having the plunger beneath the fixed sieve. Flap valves in the plunger and also in the bottom of the hutch where the water enters, eliminate the suction effect of the usual type of fixed-sieve plunger jig.

A. W. Gauger,† State College, Pa. (written discussion).—It is regrettable that a relationship between ore dressing and coal preparation has been assumed because nearly all coal-cleaning processes involve adaptations of ore-dressing machinery, whereas the problem is really very different in the case of coal.

Little or no information on the inorganic minerals present in coal is available in the literature. All conclusions seem to have been based on ash, an inorganic material that does not exist in the original coal. Fortunately pyrite in coal can be readily recognized.

While coal cleaning has had to begin at the face underground and will continue to do so as long as hand loading continues, the fact that mechanical loading in bituminous coal mines east of the Mississippi increased some 350 per cent between 1923 and 1931, although even in 1931 only one-eighth of the total was machine loaded, indicates that mechanical preparation must increase and hand preparation decrease with the trend in mechanical loading.

I am convinced that the problem of coal preparation requires fundamental study with regard to the properties of coal substance and its organic mineral matter. This must include such factors as degradation, grindability, and the fundamentals of drying, as well as the physical principles underlying the separation of the coal substance from its inorganic mineral matter.

J. Griffen,‡ Pittsburgh, Pa. (written discussion).—The relative values of the products have had a greater influence on the development of the two arts of ore dressing and coal preparation than has been fully appreciated. Ore dressing came first and was developed for the concentration of minerals that had relatively high values per ton, so that elaboration of flow sheet was economically admissible and high efficiencies were required of the various units to yield highest recovery in values consistent with proper quality of products. Scientific study contributed to the perfecting of the processes, particularly classification and jigging.

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With the development of low-grade porphyry copper deposits came large mills with simple flow sheets and low operating costs, culminating in the fine-grinding all-flotation plants of today.

When coal washing followed ore dressing the economic factors involved called for simple, cheap and effective devices. The result was the Baum jig and launder washers, such as the Blackett, Elliot, Bangest and Wunderlich. None of the latter attained wide commercial success although satisfactory in many specific instances. Each lacked the quality of universally giving uniform performance.

The development of Rheolaveur principles by Antoine France and his staff overcame the irregularity in operation. This device uses the complementary principles of launder stratification and hydraulic classification in a deep bed of material flowing in a trough. The essential feature is the thick intermediate regulating stratum between the pure coal layer above and the refuse layer below. This regulating stratum is recirculated and since it consists largely of intermediate gravity material between coal and refuse it forms a barrier between the two and also absorbs the fluctuations in quality of feed so that variations in quality of products do not ensue. The simplicity, high capacity and uniformity of operation of Rheolaveur units makes them attractive for application on ores and success has already been met in washing iron ore substantially minus $\frac{3}{16}$-in. size.

Experience shows that it is unsafe to prejudge the possibilities of a machine in a new field. It was long argued that tables used on ores had too low capacity and too high operating costs to allow their use on steam sizes of anthracite, but when actually tried it was found that table capacity on coal was much greater than on ore and, furthermore, much larger sizes could be treated, so that today tables are standard practically on pea size of anthracite through $\frac{13}{16}$ and over $\frac{9}{16}$-in. round holes and tables even clean flat chestnut through 2 by $\frac{7}{16}$-in. slotted screen and on $\frac{13}{16}$-in. round-hole screen.

To sum up, coal washing has drawn heavily on ore-dressing experience in the past, but the generally lower value of coal has forced coal washing to strike out into new paths, and developments in both arts should be carefully followed with an open mind and with the assurance that each may profit from the experience and technique of the other.

T. W. Guy, Charleston, W. Va. (written discussion).—Dean Holbrook compares the fairly fixed chemical composition of specific minerals and rocks with the complex organic chemical nature of coal and points out that today there is no acceptable standard as to what constitutes clean bituminous coal, although the anthracite industry has a generally accepted standard of purity.

Economic considerations warrant the cleaning of coal as well as ores but the separation should never go beyond the point where it no longer increases the net value of the product for its intended use. No refined metal is supposed to be pure and failure to frankly recognize that the same is true of coal is causing heavy economic waste.

The important balance between costs and values due to separation of impurities is simpler with ores and more complex with coal, largely because the lack of definite bases for coal values makes it difficult to measure and prove the cash value of coal cleaning or improved preparation. It is a question how much of this indefinite classification and valuation of a given coal is due to lack of uniformity in the coal itself as shown by variations in analyses, and how much is due to errors in sampling or measuring. It is understood that the A.S.T.M. is studying this condition.

The value of steel and many other materials and alloys is tremendously affected by slight variations in percentages of impurities present, so that it has been necessary to develop standard methods of determining these impurities and their exact effects on the properties of the material.

The A.S.T.M. and the A.S.A. have set forth standard specifications for sampling coal which are designed principally for run of mine to give an accuracy within 1 per cent of the true ash content. These specifications also call attention to the various factors and variables that affect the accuracy of sampling and provide for a larger or a smaller minimum standard weight of sample for a very substantial change in size and quantity of impurities, but it is believed that too much is left to the judgment of the sampler.

Finally, it is to be concluded that: (1) with much of the well prepared coal on the market today, materially smaller samples than the A.S.A. standard will give the required degree of accuracy, while under certain circumstances materially larger samples and special precautions will be necessary to secure the desired accuracy; (2) that the standard should specify definite procedure and quantity of sample for each typical size, for different percentages of refuse, and for different grades of coal, so that these variables may be at least approximately taken into account when the sample is taken and being prepared; (3) the expected accuracy should be specified for each standard sampling, and also the necessary changes to give greater accuracy when circumstances justify it; (4) the accuracy and cost of a sample must bear a reasonable relation to the value measured.

Such extensions of the standards should be made only after additional research and comprehensive study of the variables involved and the final result should be to put both consumer and producer in a position to measure and evaluate both preparation and inherent values in coal to better advantage.

H. F. Hebley,* Chicago, Ill. (written discussion).—While the older art of ore dressing has exerted a great influence on coal preparation, many modifications and departures have been necessary in order to treat coal efficiently.

Rigid, regular sampling for control of operations is just as essential in coal preparation as in ore dressing. Standard technique and the use of hand and automatic methods are applicable in both cases although on coal it is often rather difficult to arrange an automatic sampler that will make a representative cut of the stream flow of the product to be sampled. The preparation of samples in the hands of the samplers of both industries run parallel until that point of divergence is reached when the vastly different information required regarding the two materials under discussion makes different treatments imperative.

For settling sludge the well-known Callow cone has its counterpart in the huge cones commonly used in coal washeries abroad. A recent improvement consists of nesting several cones inside each other with a clearance of a few inches between adjacent walls. Solids tend to settle to the inside wall of each cone and slide down to the apex, the movement being hastened by the convection currents set up. Various coagulents such as starch and potassium xanthate have been used to hasten the settling effect.

Screening of fine coal below ⅛ in. is difficult, especially if a critical percentage of moisture or any coagulent material such as clay is present to cause blinding. Under such circumstances vibrating screens are better than any other screens but even these are not very satisfactory in sizing as fine as 35 or 48 mesh.

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* Allen & Garcia Co.

For this reason dedusting by air aspiration is growing in favor. On a West Virginia coal ½ in. to 0 in size, with 10 per cent free moisture, it was desired to make a separation at 14 mesh by aspiration. The results are shown in Table 1.

Another test on the same coal showed equally good results when the separation was made at 48 mesh instead of 14 mesh.

Sometimes a coal with a fairly high ash content and fusion point may yield a cleaned coal product with lower ash-fusion temperature because of the removal of a high-melting impurity in the cleaning. The slagging of ash (clinkering) and the fly-ash nuisance in boiler settings approach closely the problems of slag formation encountered in metallurgy. The Bureau of Mines, the University of Illinois and the Carnegie Institute of Technology have studied the action of chemical compounds (mostly chlorides of the alkali metals) to prevent corrosive action of flue gases, the slagging difficulties from fly-ash on boiler tubes and walls and the clinker nuisance on grates, but, as far as the writer knows, no investigation has been made on the "chloridizing roasting" action on iron pyrite present in coal. This information would be a distinct aid to consumers of middle western fuels with their high sulfur content and low fusion-point ash.

TABLE 1.—*Separation by Aspiration*

Size	Percentage			Per Cent Removed[a]
	Raw Coal	Dedusted Coal	Dust	
− ½ in. + 4 mesh..........	21.2	29.8	0.0	0.0
− 4 +14 mesh..........	41.8	55.3	5.9	3.5
−14 +48 mesh..........	24.0	12.8	51.0	60.5
−48 mesh....................	12.0	2.1	43.1	88.8
Total......................	99.0	100.0	100.0	

[a] This shows the percentage of the material of each size in the raw coal that went into the dust product.

D. F. IRVIN,* New York, N. Y. (written discussion).—Gradual trial and adoption of metallurgical methods in coal preparation are shown in the picture of evolution of practice drawn in Dean Holbrook's paper. It is interesting and unusual that the first problem attacked on dewatering concerned a less valuable product, the fine coal sludge. Perhaps this was because slime-settling devices had been well perfected for ore slime. At any rate, Dorr thickeners were installed 10 to 15 years ago to settle sludge, to clarify water for re-use and to avoid stream pollution, and in some cases to eliminate high-ash overflow slime from a thickened granular underflow. Small coal sizes have lower market value and thus offer less inducement for treatment unless through specially low ash content and by blending with other sizes the producer can build up a specialized market outlet.

About the time Dorr thickeners started on coal in this country some sporadic trials of Oliver filters occurred in England. Centrifugal drying began in 1913 or 1914 when the Elmore drier was tried out by the Pennsylvania Steel Co. However, filtration was not actively resumed until 1929, when it was installed on Rheolaveur fines around Pittsburgh. At the Champion No. 1 washery of the Pittsburgh Coal Co. trials were made of Laughlin, Dorr, Oliver, Genter and American filters, leading up to the present plant where Oliver filters handle fine product, including some flotation concentrate, at the rate of 70 to 80 lb. dry-weight coal per square foot of filter area per hour. Feed has about 50 per cent moisture and the filter cake 21 to 24 per cent. Sizes are 12 per cent + 48 mesh, 28 per cent −48 + 100 mesh, 25 per cent −100 + 200 mesh and 35 per cent −200 mesh. Subsequent installations are Genter filters at the Clairton plant and American disk filters at the Nemacolin plant of the Buckeye Company.

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* Oliver United Filters, Inc.

For coarser coal, tests on the filter at Champion No. 1 showed that different methods must be applied. S. M. Parmley's paper at the September, 1930, meeting of the A. I. M. E. covers rotary drier operations at Champion No. 1 on Oliver filter cake and minus ⅜-in. product from centrifuges. On these coarser sizes the next step was the use of the straight-line Dwight-Lloyd drier in two Indiana washeries to dehydrate minus ¾-in. washed coal. This application of the principle of the Dwight-Lloyd sintering machine to drying is proving more economical than other methods and it also avoids degradation. Producing dry coal, it renders unnecessary the continuation of the practice of adding calcium chloride solution in winter shipments of washed coal to prevent freezing in the cars.

Heat drying of coal over 1 in. in size was considered uneconomical but following experience obtained in drying mixed coarse and fine ore the Dwight-Lloyd drier was tried on a coarse and fine coal mixture and it was found that the presence of coarse pieces greatly assisted the drying of the fine, while the larger sizes themselves underwent an easy drying by the heated air current.

Coal drying has thus expanded from slimes to fines and thence to coarse and coke may now be included. The Dwight-Lloyd drier, with its nondegrading quality, has also a field in the drying of damp or wet raw coal in plants employing dry preparation methods.

The exact limitations of coal dewatering and drying form a fluctuating boundary determined by market conditions, and influenced by current progress in technical developments within the industry.

D. R. Mitchell,* Urbana, Ill. (written discussion).—While coal (speaking principally of bituminous coals) is a complex organic substance, the mineral impurities associated with it are usually fairly uniform in type and in their mode of occurrence and distribution for localized areas. It is of paramount importance for the coal user

Table 2.—*Variation in Percentage of Ash*

Sample No.	Head Sample, Ash, Per Cent	1.40 Float	
		Weight, Per Cent	Ash, Per Cent
1	8.9	89.3	4.9
2	8.7	90.5	4.9
3	11.8	86.0	6.1
4	9.8	86.7	5.5
5	10.1	88.7	5.4
6	11.3	85.5	5.9
7	9.6	88.3	5.1
8	8.7	90.5	5.5
9	19.0	65.2	5.8
10	19.0	68.3	5.5

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[*] Assistant Professor, Department of Mining and Metallurgical Engineering, University of Illinois.

to recognize the difference in physical and chemical properties of coal, but the problem of removing objectionable mineral matter need not be clouded because information is lacking as to the exact nature of the organic compounds making up the coal substance.

In concentrating a copper ore containing chalcopyrite ($CuFeS_2$) the ore dresser is interested only in recovering the chalcopyrite itself and not in the separation of copper, iron and sulfur. Similarly, the coal-preparation engineer should be interested only in the removal of harmful impurities that are separable and should have no concern over separating coal into its elements of carbon, hydrogen and oxygen.

Differences in physical and chemical properties of coal, as fired or otherwise utilized, mined in the same district are often due to the presence of undesirable impurities and not so much to difference in the coal. This conclusion is based on the fact that the pure coal substance obtained by float-and-sink separations shows uniform chemical analysis and uniform B.t.u. values over wide areas. Table 2 shows that while 10 samples of coal 6, from mines in Franklin County, Illinois, gave ash contents varying from 8.7 to 19.0 per cent, the portions floated at 1.40 sp. gr. varied only from 4.9 to 6.1 per cent. Exceptions may occur where geological agencies such as igneous intrusions change the coal substance locally.

Technologic advances in coal preparation in recent years have been along the lines of mass treatment in gravity concentrating devices both wet and dry, together with improvement in screening and mixing practices. High unit capacity has been stressed more than efficiency of the individual machine. Advances in ore dressing have been mainly along crushing, classification and froth-flotation lines, which find little counterpart in coal-preparation practices. Also, more attention is paid to efficiency of individual pieces of equipment. The ore-dressing engineer has available a far greater fund of published engineering and economic data than the coal-preparation engineer.

Dean Holbrook has referred to the lower unit value of the product of coal mines as compared with metal mines to account for the higher scientific excellence of the ore plants over the coal-preparation plants. This has been a convenient alibi to cover up shortcomings and is not in accord with actual facts. The Miami mill in Arizona treated copper ore in 1930 worth net about $1.43 per ton. Milling costs were 29.671 cents per ton, or 20.8 per cent of the value of the ore. The Alaska Juneau mill treated gold ore in 1929 valued at 92 cents per ton. Milling costs were 32 cents per ton equal to 34.8 per cent of the value of the ore. Certainly, coal cannot be valued at any lower figure for the years given. If coal is valued at $1.50 per ton and 10¢ is spent in cleaning and preparing it for the market, only 6.7 per cent of its value has been used for beneficiation purposes.

Although comparisons on such widely separated types of mineral preparation may be shown to be fallacious, yet the general principles hold true. Numerous examples exist of low-grade or complex ores which are being worked at a profit and which show an amount expended for beneficiation purposes in relation to the ore much above that usually considered a maximum by coal operating companies. It would seem, therefore, that the coal-mining and preparation fraternity have considerable to learn from technicians working on ores. It is not to be inferred that there are no efficient coal-preparation plants. There are many.

It seems that there is a gap that needs to be bridged between advances made in coal utilization and coal preparation in the general field of marketing. Adoption of technical improvements in some respects has been held back by a lack of similar advancement in marketing and servicing in conjunction with coal preparation. It is in this particular field that the coal-preparation engineer faces his greatest problem.

R. E. Rightmire, Fairmont, W. Va. (written discussion).—The subject has been well covered generally, by the author, in presenting a clear picture of the two processes. The phrase "mechanical preparation of coal" would appear preferable to "coal preparation" in the title, since there is a great deal of hand cleaning of coal by picking at the working face, at the tipple and on railroad cars. This hand cleaning of some coals is accomplished practically entirely in the mine by means of the mining methods adopted. There is little counterpart or relationship of this hand cleaning in ore dressing.

In the mechanical preparation of coal is found the close similarity to ore dressing and the tonnage of bituminous coal so cleaned for marketing is constantly on the increase in recent years. Sizing of coal is largely to meet conditions of marketing while in ore dressing it is essential for highest efficiency of the concentrating machines.

The existence of a standard of purity for anthracite and lack of such standard on bituminous coal is explainable by the fairly close similarity of chemical and physical character of anthracite as compared to the wide range of characteristics of bituminous coal. This situation does not look promising for the establishment of one generally acceptable standard for purity of bituminous coal.

Coal cleaning has the primary purpose of improving analytical and B.t.u. value and under the impulse of higher freight rates and more critical demand by the consumer, the prospect is that more and more bituminous coal will be prepared mechanically both for industrial and metallurgical uses. Recently there have been several instances of mechanical cleaning being seriously considered for coals of excellent analytical quality, with the main objective of improving the fusion temperature of the ash.

I. M. Symonds, Pottsville, Pa. (written discussion).—Experimental engineers in ore dressing and coal preparation have the same objectives: improvement of results, control of operations, and cutting costs without injury to operations. More progress has been made along these lines in ore dressing than in coal preparation, because of certain fundamental reasons. As Dean Holbrook says, Dr. Richards was transforming ore dressing from an art to a science when coal preparation was being born. As yet no one has come forward to render a similar service to preparation. A thorough organization of known facts would be of great value.

The first step in improvement is to know what present results are. Metallurgical results in a mill are easily determined by assays on a few representative samples. It is daily routine to estimate the valuable mineral saved in the concentrate and lost in the tailing. On the other hand, the preparation men have been working for years with the only data available along this line being on the tonnage of prepared coal loaded. Today some of the operators are taking daily or even hourly samples of the refuse for analysis by sink-and-float or hand-picking methods. Such steps have greatly aided the preparation men in determining present results and controlling operations.

There are reasons why the millman has progressed further in determining his results. Sampling a mill's finely ground head, concentrate and tailing is much easier than sampling the large lumps in a breaker's feed and refuse and the fairly coarse material in the product. Samples, to be representative, must be much larger in preparation, but even after obtaining a good sample there is much more work to be done. All samples are sized, usually by hand, and each size worked separately. In some cases ash analysis can be used where the assay is used on mill products, but often, as in the coarser sizes of anthracite, hand picking is used. Sink-and-float analysis has replaced some hand picking, but will never do it entirely in anthracite as long as appearance is such a factor in the market.

In conducting experimental work in a large mill the engineer is greatly aided by the mill being divided into independent sections. Even radical experiments may be run, because only a small portion of the total tonnage is involved. It is common to have at least one section on which experiments are regularly conducted. The average breaker is built as a unit, or at the most in two sections. Any experiment may have a detrimental and expensive effect on the entire production. The result is that many tests that should be run in the breakers are not run, or are run for too short a period to be of much value.

The millman has found laboratory equipment of much value in the control and improvement of operations. The preparation man has found no comparable aid, although much can probably be done in small-scale testing of fine coal. No inexpensive laboratory machine can furnish the data on preparing coarse coal as a 1000-gram flotation machine will on a floatable ore. To stay within practical limits preparation men must rely in many cases on opinion and judgment, while the ore dressers have facts. Human judgment being fallible, the millmen have the advantage.

Use Classification of Coal for Stationary Steam Generation

By Thomas W. Harris, Jr.,* Wilmington, Del.

(New York Meeting, February, 1933)

This paper was written in conjunction with the work being undertaken by the Technical Committees on Classification of Coal, and an effort has been made to compile information relative to the use of coal for stationary steam generation in order to use this information to assist the Committee on Classification of Coal for the ultimate purpose of elimination of losses both by the reduction of individual effort and waste in the consumption of material.

Under the auspices of the National Association of Purchasing Agents, a questionnaire was drawn up to develop this information from stationary steam plants throughout the United States in order to consolidate the information developed from the many hundreds of boiler tests on coals consumed by various coal-burning equipment. These questionnaires covered:

1. The company reporting, location and method of delivery to the plant.
2. Complete coal data, including description of coals used (with sufficient space to include first, second and third preference) on source of coal used, complete analysis and factors affecting choice of coal used.
3. Plant data, divided into:
    (*a*) hand-fired,
    (*b*) overfeed stokers,
    (*c*) underfeed stokers,
    (*d*) traveling grate stokers,
    (*e*) pulverized coal;

and included complete description of furnace data, draft data, coal-burning rate, type of stoker, type of pulverizer, efficiency data and remarks.

The questionnaire was drawn up by consultation with various combustion engineers, including B. J. Cross, Test Engineer of the Combustion Engineering Co., W. H. Fulweiler, of the United Gas Improvement Co. and the U. S. Bureau of Mines. In all, 11,538 copies of the questionnaire have been distributed, of which 4369 were sent in duplicate.

Our estimate of the total volume of bituminous coal consumed in stationary steam boilers over the 12 months of 1930 is approximately

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* Division Purchasing Agent, E. I. du Pont de Nemours & Co., Inc.; Chairman of the Subcommittee of the Technical Committee on Use Classification, and Chairman of the National Coal Committee of the National Association of Purchasing Agents.

125,000,000 net tons, which is in the neighborhood of 30 per cent of the bituminous coal consumed in the United States over the same period. This 125 million net tons is not quite complete, as it does not include a relatively small percentage of coal consumed in stationary steam plants by railroads, by product and beehive coke plants, coal-gas plants or any apartment houses and hotels, but does include primarily the plants where efficient generation of coal is practical.

Of this total estimate of coal consumed under stationary steam boilers in the United States, returns on the questionnaire amount to 15.5 per cent.

According to a report of the Institute of Economics in Cooperation with the United States Geological Survey, on Sources of Coal and Types of Stokers and Burners Used by Electric Public Utility Power Plants, compiled by William Harvey Young, which covered the year 1928, the following information was obtained:

Total net tons consumed by public utilities in the United States was approximately 42 million, of which the percentages of coal consumed in the various types of equipment were as follows:

	Per Cent
Hand-fired boilers	2.3
Overfeed stokers	3.0
Underfeed stokers	49.0
Traveling-grate stoker (chain grate)	28.7
Pulverized coal	17.0

According to consuming areas, this coal consumed by public utilities was subdivided as follows:

	Per Cent
New England	6.70
Middle Atlantic	33.00
Ohio	10.00
Southern Michigan	5.70
Illinois and Indiana	20.40
Lower Missouri Valley	6.80
Lake Docks Territory	4.00
Southeastern	8.70
Southwestern, Rocky Mountain and Pacific	4.70

The public utility plants were divided into the following classification, based on the number of tons consumed per year, giving the percentage of tonnage consumed:

	Per Cent
Plants consuming	
Less than 2,000 tons	0.3
2,000 to 10,000 tons	4.1
10,000 to 50,000 tons	12.9
50,000 to 100,000 tons	13.0
100,000 to 200,000 tons	16.7
200,000 to 300,000 tons	17.0
300,000 and over	36.0

In the above data furnished, the study included 842 plants out of a total of 1168, and as the 326 plants not included consumed only 1.5 per cent of the coal consumed by public utilities, most of these plants omitted burned less than 2000 tons per year.

From the information available in the Purchasing Agents Association, stationary steam plants, other than public utility, showed a larger percentage of coal consumed in the smaller consuming groups and a smaller percentage in the larger consuming groups as compared with public utilities.

The percentage of replies on the questionnaire corresponds relatively closely to the percentage of consumers in the United States falling into the consuming classifications mentioned above, so that the replies are fairly representative of the varying steam conditions based on the amounts consumed.

G. B. Gould, president of the Fuel Engineering Co. of New York, cooperated to a very large extent in making the survey on the returns on the questionnaires, and this survey brought forth the following results:

The various coal-consuming localities were grouped according to those divisions of the coal fields to which they are naturally attached by existing transportation facilities and rates, as it was obvious that such information as was supplied by these questionnaires submitted by plants scattered across the whole industrial area of the United States would have to be interpreted in relation to the type of coal naturally available to each one.

The questionnaires definitely showed that the location of a coal-burning plant automatically determines for that plant certain portions of the coal-producing regions, from which it is practically compelled to secure its coal, because of the preponderant effect of transportation costs. The questionnaires were, therefore, grouped according to seven principal divisions of the industrial coal-consuming market, six of which are shown on the accompanying map (Fig. 1). These seven divisions may be designated for convenience as follows: (1) Eastern New England, (2) Middle Atlantic, (3) Pittsburgh, (4) Central, (5) Mid West, (6) Upper Lakes, (7) South Atlantic (Not shown on map. Includes Virginia, North and South Carolina, eastern portion of Kentucky and Tennessee).

Outside of this industrial area, there are a number of outlying coal fields, including those that are situated in Alabama, Missouri, Kansas, Oklahoma, North Dakota, Utah, Colorado, New Mexico, Wyoming, Washington and Texas. With the exception of the Alabama coals, which are generally distributed over several of the southern states, the consumption of coal from these other isolated fields is very largely confined to the state within which the coal is mined. The entire amount of coal mined in all of these districts is a relatively small proportion of the total production of the country and the tonnage reported from these districts on the questionnaires is out of proportion on the high side.

The boundaries between these seven sections of the market are, of course, not capable of being fixed with great precision. In many cases, the division is more in the nature of a neutral zone, within which the coals from two different sections of the coal fields compete with each other on more or less equal terms.

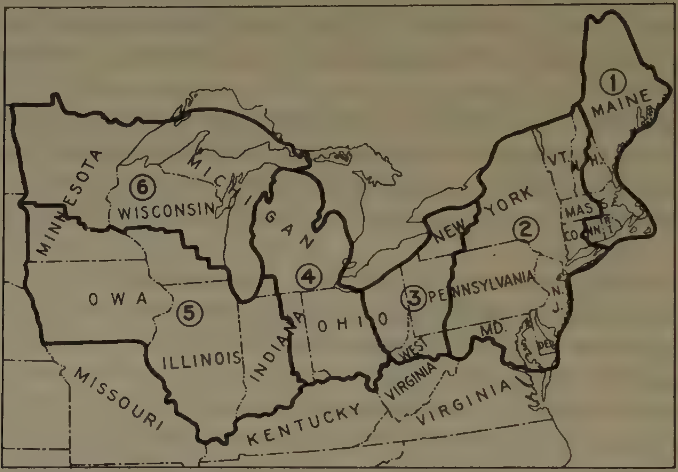

FIG. 1.—PRINCIPAL DIVISIONS OF INDUSTRIAL COAL-CONSUMING MARKET. Reproduction of base map licensed under copyright by Rand, McNally & Co.

A.—COAL SELECTION CONTROLLED BY GEOGRAPHICAL LOCATION

How nearly the choice of coal adheres to the controlling economic factor of transportation may be seen from the tonnage reported from each market division, classified according to the sources of coal used as shown on Table 1.

TABLE 1.—*Tonnages Reported from Market Divisions*

Market Division	Pennsylvania and Maryland Low and Medium Volatile Bituminous and Anthracite	Pennsylvania, Northern West Virginia and Ohio High-volatile	Southern West Virginia Low-volatile	Virginia, South West Virginia, Eastern Kentucky High-volatile	Illinois, Indiana and Western Kentucky	Other Fields Serving Local Markets
Eastern New England.			554,000			
Mid-Atlantic.........	4,160,000	167,000	395,000			
Pittsburgh...........	223,000	3,305,000				
Central..............		128,000	109,000	3,048,000		
Mid-West............		70,000	53,000	7,000	3,861,000	
Upper Lakes..........		196,000		154,000	10,000	
South Atlantic........			941,000	200,000		
Outlying markets.....						1,955,000
	4,383,000	3,866,000	3,052,000	3,409,000	3,871,000	1,955,000
Total... 19,313,000						

It will be seen that by classifying the plants according to the natural market divisions, in one case 83 per cent, in another 87 per cent, and in the other five over 90 per cent of the coal reported was secured from the coal field nearest to it, and involving the least cost of transportation. The small remainder reported from other districts is accounted for by plants at the borderline between two market divisions.

B.—SIZE OF PLANTS REPORTING ABOVE THE AVERAGE

The size of the plants reporting has, of course, an important bearing on the interpretation that is placed upon the data presented. This may be most easily expressed in terms of annual coal consumption. The average of all plants reporting both coal consumption and type of equipment, by classes of equipment, is as follows:

	Tons per Year		Tons per Year
Hand fired	8,600	Chain grates	47,000
Overfeed stokers	9,600	Pulverized coal	118,000
Underfeed stokers	45,000		

These figures are somewhat misleading because of the large influence on the average of a small number of very large plants. Slightly less than 10 per cent consume 100,000 tons or more a year each and account in the aggregate for 58 per cent of the tonnage reported, all in the underfeed-stoker, chain-grate and pulverizer groups. Excluding this small number of very large plants, the more nearly representative majority consume, on the average, the following amounts:

Underfeed stokers, 22,000 tons per year; chain grates, 22,000; pulverized coal 38,000. Even corrected in this way, these figures indicate that the great majority of plants reporting are among the larger plants. Indications and conclusions reached from this survey, therefore, mainly represent the larger plants. What effect this has on the results it is difficult to say, but at least these plants should be among the more discriminating in the matters of coal selection.

C.—SIZE OF COAL USED BY PLANTS REPORTING FROM EACH DIVISION OF MARKET

Next to the cost of transportation, the most important determining factor in coal selection for steam generation, as shown by the reports, on a large scale is the size of the coal, not from the standpoint of engineering advantages in use but because of the fact that in the evolution of our use of coal, certain byproducts, or residual small sizes have resulted, which customarily sell at prices enough lower than the lump or run of mine coal to make them attractive to plants equipped and operated to use them efficiently. To what extent the larger steam plants have adapted themselves to these residual sizes is illustrated by the analysis as shown by Table 2.

TABLE 2.—*Sizes of Coal Used*

Market Division	Slack or Screenings		Run of Mine		Stoker and Nut		Total Tons
	Tons	Per Cent	Tons	Per Cent	Tons	Per Cent	
New England	526,000	95	28,000	5			554,000
Mid-Atlantic	827,000	23	2,606,000	74	104,000	3	3,537,000
Pittsburgh	2,277,000	65	916,000	26	335,000	9	3,528,000
Central	3,193,000	97	18,000	1	74,000	2	3,285,000
Mid-West	3,941,000	99			50,000	1	3,991,000
Upper Lakes	360,000	100					360,000
South Atlantic	1,140,000	99	1,000	1			1,141,000
Outlying markets	1,589,000	98	25,000	2			1,614,000
	13,853,000	77	3,594,000	20	563,000	3	18,010,000
Total, excluding Mid-Atlantic	13,026,000	90	988,000	7	459,000	3	14,473,000

The noticeable fact here is the very large proportion of slack or screenings, reported from all except the Middle Atlantic division of the market. The relatively large proportion of run of mine shown by the Middle Atlantic division is readily explained by the fact that most of the coal used in this area comes from the low-volatile and medium-volatile fields of Pennsylvania and Maryland, in which only a very small amount of slack is produced.

D.—AMOUNT OF COAL REPORTED FOR EACH TYPE OF EQUIPMENT

The distribution of the tonnage reported, according to the type of coal-burning equipment, is shown in Table 3.

TABLE 3.—*Tonnages for Different Types of Equipment*

Market Division	Hand-fired		Overfeed		Underfeed		Chain Grates		Pulverizers	
	Tons	Per Cent	Tons	Per Cent	Tons	Per Cent	Tons	Per Cent	Tons	Per Cent
New England	36,000	6			347,000	63			167,000	30
Middle Atlantic	153,000	3	80,000	2	2,716,000	63	969,000	22	461,000	10
Pittsburgh	18,000	1	29,000	1	2,215,000	62	285,000	8	981,000	28
Central	96,000	3	81,000	2	1,876,000	57	13,000		1,177,000	36
Mid-West	80,000	2	28,000	1	227,000	6	1,306,000	33	2,329,000	58
Upper Lakes			44,000	12	154,000	43	29,000	8	133,000	37
South Atlantic	13,000	1	6,000		923,000	81			199,000	18
Outlying markets	17,000	1	17,000	1	518,000	27	622,000	33	706,000	38
	413,000	2	285,000	2	8,976,000	47	3,224,000	17	6,153,000	32
Total									19,051,000	

These figures should be taken only as indicative of the type of equipment that is most frequently reported from each division, and not as a quantitatively accurate representative for steam generation as a whole. The figures do bring out, however, the fact that in plants of this size and kind (mostly industrial steam plants using 5 to 50,000 tons a year) the underfeed stoker is the principal type of coal-burning equipment, except in the Middle West, where the chain grate was widely used before the underfeed stoker had reached its present stage of development.

The use of the chain grate in the Middle West furnishes an interesting reflection of the tendency to adapt coal-burning equipment to the type of coal that, for geographical reasons, is the most economical. In the same way, the tonnage reported for chain grates in the Mid-Atlantic division is almost entirely anthracite.

In the above we endeavored to analyze and subdivide the coals consumed among the geographical divisions in accordance with the method of firing, and developed that transportation costs were a big factor in the selection of coals; that the smaller sizes of coal, on account of price, were being given preference and that in the case of the chain-grate stokers particularly the equipment had been adapted to the type of coal available. All other types of equipment are found in sections drawing from all divisions of the coal fields, therefore individual cases could be selected, if desired, to show the use of every variety of coal over the whole range of coal characteristics on any type of coal-burning equipment.

As for instance, in the Mid-Atlantic division, 93 plants specifically identified by name of mine the coal they preferred and, taking into consideration their first, second and third choice, we find that these plants picked out 91 different coals.

In other words, it appears that the selection of coal for steam generation as it is now practiced is not a matter for classification of either coal or equipment, but one of individual adaptation to the particular location, operating requirements and coal-market conditions, peculiar to each plant at any given moment.

This probably is due to the fact that although there may be preferences for various coals in various types of equipment, owing to a variation in delivered costs of coal, the difference in ultimate boiler efficiency is offset by the difference in delivered price.

We next endeavored to analyze the questionnaires according to the chief factors affecting the choice of coal as requested on the questionnaires and, by tabulating these factors chosen by each plant as the first four in importance, it is possible to secure an interesting measure of the relative importance of each one, divided according to types of equipment, and sectionally. To a limited extent certain interesting relationships between the principal factors affecting choice of coal and coal quality are found. The relative importance of each factor is expressed here in terms of percentage of the whole number of plants mentioning that item among the four most important factors. The combined results for all plants are shown in Table 4.

TABLE 4.—*Percentage of Plants Indicating Each Item Among First Four Factors*

HAND AND STOKER-FIRED		PULVERIZED COAL	
1. Fuel cost	84	1. Fuel cost	76
2. Maintenance costs	66	2. Maintenance costs	76
3. Clinkering on grates	65	3. Slagging on heating surface	52
4. Slagging on heating surface	33	4. Pulverizer capacity	48
5. Labor costs	32	5. Labor costs	28
6. Overloads on boilers	27	6. Moisture	24
7. Smoke regulations	17	7. Overloads on boilers	20
8. Fluctuating loads	15	8. Fluctuating loads	20
9. Moisture	6	9. Clinkering	20
		10. Smoke regulations	16

The factor of fuel cost, which holds first place in this referendum, may be disregarded from the standpoint of its relative importance for the reason that the obvious objective of every steam plant is the lowest cost consistent with satisfactory performance of the plant, and within each plant's limitations of location and equipment.

E.—MAINTENANCE AND CLINKER FORMATION MAJOR CONSIDERATION IN HAND AND STOKER-FIRED PLANTS

The placing of maintenance at the head of the remaining factors is probably due to other effects than the *cost* of maintenance. Few steam plants, other than the large electric utilities, have any basis for making accurate comparisons of the relative effect of different coals on maintenance costs. What is reflected here is probably not so much the importance of maintenance *costs*, as the importance of maintenance *troubles*—threatened or actual interruptions to service for frequent minor repairs. These are apt to bulk large in the daily life of the operator of the typical industrial plant. Fear of them is often exaggerated as a result of some unfortunate experience in the trial and error process of coal selection.

Closely related to maintenance, as it is affected by coal quality, comes "clinkering on the grates." Higher maintenance is an almost inevitable consequence of serious clinker formation, but this, of course, is not the only reason why clinkering stands high in the list, for it also may depress efficiency, limit capacity, in extreme cases actually interrupt service, and is always objectionable to the boiler-room force. The high rating given to both maintenance and clinkering is undoubtedly a combination of the cost factor and the expression of what may be termed a "nuisance" factor.

It is probably also true that some of the votes cast for "overloads on boilers" could be considered as another way of voting for clinkering as a factor in coal selection. The ability to carry high overloads depends on the ability to maintain a relatively high rate of combustion, which in turn would be prevented by clinker formation. The scattering votes for "fluctuating loads" may reflect somewhat the same effect.

Maintenance of coal-burning equipment, as it is affected by coal characteristics, being closely related to clinker formation, and the necessity for carrying overloads being often considered the cause of clinker, it would appear that these results indicate that clinker formation is easily the main consideration, next to the obvious one of fuel cost, in coal selection for hand or stoker-fired plants. This is, of course, in agreement with common experience.

F.—Chief Factors Affecting Choice of Coal in Pulverizer Plants

In pulverized-coal plants, maintenance can be more closely related to coal characteristics. The rate of repairs on pulverizing mills is easily segregated, and may be noticeably different with different coals. The fact that the coal particles are transported at high velocity to the furnace, and much of the residual ash also passes through the boiler, economizer, air preheater and induced-draft fan, opens up possibilities of physical deterioration that may vary directly with the physical characteristics of the coal, or of its resultant ash. The high vote for maintenance of pulverized-coal plants is therefore more probably a true reflection of consideration of this factor from a *cost* standpoint.

The high place accorded slagging on the heating surfaces may be partly due to the "nuisance" as well as the cost involved. While the accumulation of slag on the heating surfaces reduces the capacity of these surfaces to absorb heat, and may increase boiler maintenance, it is difficult to measure the effect in dollars, but the operating force is distinctly conscious of the annoyance of having to keep the surfaces clean.

It is quite to be expected that pulverizer capacity should rank among the chief considerations. The capacity of the whole plant, of course, depends in the first instance upon the capacity of the pulverizing mills, and this in turn is definitely related to the grindability of the coal, a characteristic that varies over a wide range among different coals.

Among the less frequently mentioned considerations, moisture in the coal, as might be expected, ranks much higher in pulverizer plants than in plants fired by hand or stoker. Clinkering, while mentioned by 20 per cent of the pulverizer plants, is of course not the same thing as the clinkering referred to by the other plants. By clinkering is probably meant the formation of fused slag at the bottom of the furnace.

Smoke regulations apparently play a small part in coal selection, but have had some effect on the design of the plants.

G.—Factors Affecting Choice of Coal Differ According to Type of Equipment

The foregoing figures do not discriminate among the different methods of firing coal on a solid fuel bed. There are some interesting differences among them, shown in Table 5, which lists the four most important considerations, with the proportion of plants in each class.

Table 5.—*Relative Importance of Conditions in Firing*

Condition	Percentage in Various Methods			
	Hand Firing	Overfeed Stokers	Underfeed Stokers	Chain Grates
Fuel cost	86	79	82	92
Maintenance	43	71	74	55
Clinkering	61	93	68	44
Slagging		29	42	
Labor	43			
Overloads				48

Clinkering, while appearing among the first four considerations, with all types of equipment, holds a widely different position of relative importance, ranging from 93 per cent in plants equipped with overfeed stokers down to 44 per cent in the chain-grate plants. In the latter, it is subordinated by the fact that many chain grates are used for anthracite, which is less subject to clinker trouble. Maintenance and fuel cost appear also in all four classes, as a major consideration, while labor cost, as would be expected, appears only in connection with the hand-fired plants, but even in that case coal selection is considered to affect labor costs by only 43 per cent of the plants. Further reference will be made to these figures in relation to coal quality reported.

H.—Geographical Location in Relation to Factors Affecting Coal Selection

The factors affecting choice of coal, separated geographically, according to the seven coal market divisions, are shown in Table 6.

The geographical classifications of principal considerations in coal selection reveal only that clinkering and its related factor of maintenance stand at the head of the list (next to cost) everywhere, except in the Pittsburgh section, where clinkering actually heads the list by a generous margin even over the factor of cost.

While the coal-quality data that were furnished cannot be related to performance because of the incompleteness of the engineering information generally supplied, there are some interesting relationships revealed

TABLE 6.—*Factors Affecting Choice of Coal*

Market Divisions	First Factor, Per Cent	Second Factor, Per Cent
New England	Fuel cost, 100	Clinkering, 69
Mid-Atlantic	Fuel cost, 88	Maintenance, 71
Pittsburgh	Clinkering, 83	Fuel cost, 65
Central	Fuel cost, 78	Clinkering, 76
Mid-West	Fuel cost, 94	Maintenance, 60
Lakes	Fuel cost, 91	Maintenance, 82
South Atlantic	Fuel cost, 75	Maintenance, 75
	Third Factor, Per Cent	**Fourth Factor, Per Cent**
New England	Maintenance, 62	Slagging, 38
Mid-Atlantic	Clinkering, 67	Slagging, 27
Pittsburgh	Maintenance, 57	Slagging, 52
Central	Maintenance, 65	Slagging, 43
Mid-West	Clinkering, 51	Labor, 36
Lakes	Clinkering, 55	Overloads, 36
South Atlantic	Clinkering, 50	Labor, 50

between type of equipment and chief factors influencing choice of coal on one hand and reported fusing temperature of the ash on the other.

I.—REPORTED PREFERENCE WITH RESPECT TO FUSING POINT OF ASH ACCORDING TO TYPE OF EQUIPMENT

Slightly more than one-half of those submitting reports gave a figure for the fusing temperature, and these figures have been classified both according to types of equipment and according to the seven market divisions. While there is reason to doubt the strict accuracy of many of these reports in respect to coal quality, a matter that will be discussed more fully at another point, such inaccuracies are not of sufficient weight to distort the general conclusions which may be drawn from the reports, taken in the mass. The fusing-point data, classified according to type of equipment, given on the basis of the percentage of each equipment group using coal classified within each of four divisions of the fusing-point range, are shown in Table 7.

TABLE 7.—*Fusing Temperatures*

Fusing Temperature of Ash, Deg. F.	Percentages with Different Methods					
	Hand Firing	Over-feed Stoker	Under-feed Stoker	Chain-grate Stoker[a]	Total Solid Fuel Bed Equipment	Pulverized Coal
Under 2300	0	7	10	58	12	40
2300 to 2499	8	0	16	34	15	20
2500 to 2699	40	27	32	8	31	23
Over 2700	52	66	42	0	42	17

[a] Bituminous coal only.

It will be noticed at once that in connection with the hand-fired plants and those equipped with either overfeed or underfeed stokers, there is evidence of a distinct preference for coal having a high fusing temperature of the ash. This is even more evident in the geographical classification of these data, in sections of the market where a wide range of choice, in this respect, is possible.

The high percentage of plants reporting fusing point below 2300° F., among those equipped with chain-grate stokers, should not be taken as an indication of preference for coals in the lower range of fusing point. In the geographical distribution of different types of equipment, it was shown that the plants equipped with chain grates and burning bituminous coal were almost exclusively in the Middle West, and thus were situated in a section that draws its coal almost entirely from the Illinois and western Kentucky fields, where the fusing point is characteristically low. These figures for the chain-grate plants are not, therefore, an evidence of choice, but rather of necessity. Viewed in another light, however, the necessity may have something to do with the highly localized preference for chain grates.

The higher proportion of pulverized-coal plants reporting low fusing point may be taken not so much as an indication of positive preference for such coals but rather as a tolerance of them. The pulverized-coal plant, free from the devastating effect of serious clinker formation in the solid fuel bed, is often able to use the coal of lower fusing point without serious detrimental effect on performance, and to buy such coals at a price advantage, which in no small measure arises from the preference that exists among the stoker-fired plants for the coals of higher fusing point. In the special case of the slag-tap type of pulverized-coal furnace, there exists, of course, a definite preference for the low fusing point. These plants, however, are few in number, and only one is found among those reporting in this investigation.

J.—RELATION BETWEEN FUSING POINT, CLINKER FORMATION AND TYPE OF EQUIPMENT

There is an interesting agreement between the figures on the fusing point of the ash and the relative importance attached to clinkering as a factor in the choice of coals, arranged according to types of equipment. The overfeed stoker plants show the strongest preference for coals having a fusing point above 2700°, 66 per cent of them reporting coals in this class. Reference to the analysis of factors affecting the choice of coal shows that the overfeed stoker plants constitute the only class of equipment that puts clinkering in first place, 93 per cent of these plants mentioning this as one of the principal considerations.

The hand-fired plants put clinkering second only to fuel cost, and this type of plant shows the next strongest preference for coal above 2700° in fusing point, more than one-half of them reporting coal in that class, and, all together, 92 per cent reporting coal with a fusing point of above 2500°.

The underfeed stoker plants, as a class, put clinkering third, more than two-thirds of them mentioning clinkering among the chief factors affecting the choice of coal, and 42 per cent of this class of plants report coal above 2700°.

In contrast to these, the chain-grate plants put clinkering fourth, only 44 per cent of them mentioning this as a factor in coal selection; 58 per cent of these plants report the use of coal below 2300° in fusing point and 92 per cent use coal which they rate below 2500°.

On the other hand, there are numerous specific cases that demonstrate apparent lack of knowledge of the relationship between coal characteristics and plant performance. Several plants report clinkering as a major consideration in the choice of coal, and also definitely state that the fusing point is either "not known" or "not determined." A still larger number, though of course a small minority of the whole number, report clinkering of major importance, and at the same time report the use of coal having a fusing point in the neighborhood of 2300°, though these plants were all situated in an area where coal of much higher fusing point is easily obtainable. Three plants, all operated by the same company, all indicating clinkering as important, and all using anthracite, report the fusing point as 2300°, although it would take a very careful search of the anthracite fields to find any coal having a fusing point anywhere near so low a point.

Another plant reports clinkering as its first consideration, next to cost, and gives 2375° as the fusing point of its first choice of coal, and 2800° for its second choice.

K.—Fusing Point of Coal Reported in Use, According to Principal Coal-market Divisions

Classified geographically, according to the seven market divisions, the fusing point reported for the coal in use, given in percentage of plants, is shown in Table 8.

TABLE 8.—*Fusing Point for Coal in Use*

Fusing Point, Deg. F.	Percentage of Plants							
	Eastern New England	Mid-Atlantic	Pittsburgh	Central	Mid-West	Lakes	South Atlantic	Total All Sections
Under 2300......	0	5	20	2	74	10	10	15
2300 to 2499.....	6	14	27	11	15	30	6	15
2500 to 2699.....	38	14	43	39	11	40	56	29
Over 2700.......	56	67	10	48	0	20	28	41

In a rough way these proportions correspond to the fusing-point characteristics of the sections of the coal fields from which each one of these market divisions secures its coal, with one exception, which will be discussed below. They also reflect the already established preference for coal in the higher fusing-point ranges, within the limits of choice imposed by the typical characteristics of the coals available in each section.

L.—Modifying Considerations with Respect to Reported Fusing-point Data

There is a tendency for figures of this kind to show a higher percentage in the higher fusing-point brackets than a strictly accurate and typical cross-section of the steam-generating demand would show. There are two reasons for this. One is that those who are successfully using a coal with low fusing point are presumably operating under conditions that make the fusing point of the coal relatively unimportant, and in such plants there is a natural tendency to ignore this coal characteristic, and consequently not to determine it. There is evidence of this in the fact that a much smaller proportion of plants in the Mid-Western section reported the fusing point than in any other section, and it will be remembered that this division reported the smallest proportion of plants considering clinker formation as a factor in coal selection.

In collecting data of this kind by the questionnaire method, there is no way of distinguishing between coal-quality records derived from the plant's own investigation and figures that are merely the passing on of sales representations, of which the latter, especially in respect to fusing point, are apt to vary somewhat on the high side.

Some allowance must be made, therefore, for this natural tendency to exaggerate the number of plants using coal in the higher brackets, in reading the figures in the last two tables. With such an allowance in mind, the figures are about what might be expected for all of the sections, except the Mid-Atlantic. In that division the figures were so far out of line with what is known to be a typical cross-section of the coal that, as a whole, moves into this area, that a test was made of them.

A list of the coals specifically identified by mine name, reported from this division, was taken off. The typical fusing point for each of these coals was independently supplied from the Fuel Engineering Co. records by the member of its staff, who had no knowledge of data contained in the questionnaires. These data were then tabulated in the same way as the data for this market division, shown in Table 8. The two sets of figures are shown in parallel in Table 9.

Further analysis of the two sets of figures revealed that slightly more than one-half the changes were caused by supplying fusing-point figures for low fusing-point coals, for which no figure was given on the questionnaires, and the remainder were caused by moving about 15 per cent of the coals, for which the fusing point was reported above 2700°, into a lower bracket. This test confirms the tendencies already described, and brings the figures for this section more nearly into line with what would be expected from a knowledge of the coal available in this area. There seems to be no way to explain why the figures reported from this division should be so much out of line, as compared with those from the other sections.

TABLE 9.—*Percentages from Mid-Atlantic Division*

Fusing Point, Deg. F.	Data from Questionnaire	Data from Fuel Engineering Co. Records
Under 2300	5	5
2300–2499	14	24
2500–2699	14	27
Over 2700	67	44
	100	100

M.—PREFERENCE FOR HIGH FUSING POINT NOT WHOLLY REFLECTION OF ACTUAL ENGINEERING REQUIREMENTS

The distinct preference for the coals of higher fusing point, which apparently is shown by these returns, should not be taken at its full face value as an indication of the actual engineering requirements of these plants. Clinker formation obviously is uppermost in the minds of a large majority of the plant operators. The effect of clinker formation is definitely undesirable. One or two unfortunate experiences, in the trial-and-error method of coal selection so widely practiced, naturally induces a fear of repetition, which causes the operator to overreach in the direction of high fusing point, as a matter of safety. Judging from the returns themselves, as well as from our own experience, the relatively high proportion of plants that report high fusing point in the higher brackets is as much a reflection of this human tendency to overreach out of fear of clinker formation as it is an indication of actual engineering requirements, determined from the combustion rate, the furnace temperatures, which have a definite relation to fusing point and clinker formation.

It is not to be overlooked that a heavy preponderance of market preference for coals having a high fusing point automatically tends to create a premium that must be paid for such coals, which are produced by a minority of the mines, even in sections where high fusing point is most commonly found. Consequently, a real or imagined necessity for coal having a high fusing point represents an economic handicap as compared with the plant that does not require it; a handicap that has for its cause either improper design to begin with or an attempt to get more work out of the plant than it was designed for, or lack of sufficiently accurate engineering discriminations as to the exact fusing-point limitation of the plant, leading to establishment of a figure for fusing point considerably above actual needs, as a matter of safety. While the supply of bituminous coal is sufficient for centuries to come, the supply of coal of high fusing point that can be mined economically is by no means inexhaustible, even from the standpoint of those who are today making investments in steam-generating plants.

Observations on Questionnaires

Certain observations can be made as a result of the survey made of the replies to the questionnaires, which can be subdivided as follows:

Pertaining to the Questionnaire Itself

1. The results of the questionnaires are based upon the fact that the great majority of the plants reporting are among the larger plants, and, therefore, should be more discriminating in coal selection.
2. Fifty per cent of the returns supplied complete analysis figures; 36 per cent incomplete and 14 per cent gave no figures at all, showing that a number of steam plants are partly or wholly uninformed as to the measurable qualities of the material they are using. Only 57 per cent attempted to indicate the factors affecting choice of their coal, which reflects an equal lack of discriminating knowledge as to why they are buying it.
3. The Middle Atlantic Division shows a tendency to report coals as having higher fusing points than they actually have.

Pertaining to the Coal Selected

1. There is no indication of a trend of certain types of coals for certain types of installations, or vice versa, with the possible exception of the chain grates in the Middle West.
2. It is a noticeable fact that there is a large proportion of slack and screenings, with the exception of the Middle Atlantic Division, consumed to the extent of practically 90 per cent among the companies reporting.

Pertaining to Consuming Equipment

1. Underfeed stoker is the principal type of coal-burning equipment except in the Middle West.
2. The use of chain grates in the Middle West furnishes an interesting reflection in the tendency to adapt coal-burning equipment to the type of coal that for geographical reasons is the most economical.
3. Smoke regulations apparently play a small part in coal selection but have had some effect on the design of the plants.

4. It is noticeable that on the overfeed stokers the strongest preference is shown for high fusing-point coal and on chain grates the greatest use of low fusing-point coals. In connection with chain grates, this is no indication of preference but an indication of coals available geographically, and bears out the possibility of the highly localized preference for chain grates.

5. Many of the overfeed stoker plants that show the highest preference for high fusing coal also show that among the many factors affecting choice of coals, it is the only class of equipment that puts clinkering first.

6. In regard to pulverized-coal plants reporting, a high percentage of low fusing coals indicates not necessarily a preference but rather a tolerance of them, with the exception of the slag-tap type of pulverized-coal furnaces.

7. Hand-fired plants show the next strongest preference for coals of high fusing point, and put clinkering second only to fuel cost.

8. Underfeed stoker plants put clinkering third among the factors affecting choice of coal, and 42 per cent of these plants reported coal above 2700° F. fusing.

9. Chain-grate plants put clinkering fourth. Only 44 per cent mentioned this as a factor for coal selection; 92 per cent used coal which they rated below 2500° fusing.

10. Because development of designs and construction of coal-burning equipment advance much more rapidly than it is possible for individual plants to keep up with, there is undoubtedly existing, as shown on the questionnaires, a marked opportunity for improvement in the steam equipment in industrial plants. However, the low prices on coal have retarded replacing of obsolescent equipment.

11. The reports do not have a tendency to establish any type of coal-burning equipment exclusive of all other types for any whole section of the coal market, with the possible exception of chain-grate preference for small sizes of anthracite. However, the reports do show that in new installations, or changes to present installations, there is plenty of room for careful study to develop the logical consuming equipment.

Pertaining to Factors Affecting Choice of Coals

1. There seem to be seven principal divisions of the industrial coal-consuming market, which are fairly well defined by transportation facilities and rates from the coal fields. The choice of coal adheres between 83 and 90 per cent to the coal field nearest it, involving the least cost of transportation.

2. Indications point to the fact that variation in delivered price of coal has been used to offset the variation in suitability of the coal to the varying consuming conditions.

3. Maintenance troubles (threatened or actual interruptions to service for frequent minor repairs) play an important part in the selection of coal in hand-fired and stoker-fired plants. However, there is indicated excessive emphasis placed on this factor.

4. Clinkering on the grates comes next to maintenance and is affiliated very closely with it, although it could also create depressed efficiency and limited capacity.

Clinkering, while it appears among the first four considerations on all types of equipment, holds a widely different position of importance, varying from 93 per cent in plants equipped with overfeed stokers, down to 44 per cent, in the chain-grate plants.

5. In connection with pulverized-coal plants, maintenance cost is the chief factor in the selection of coals and is more closely related to coal characteristics.

6. Pulverizing capacity ranked among the chief considerations and is dependent largely on the grindability of the coal.

7. Moisture ranks very much higher in consideration among the pulverizer plants than in the hand-fired and stoker-fired plants.

8. Slagging on the heating surface in pulverizer plants is given a high place in the selection of coals.

9. Labor costs appear only in relation to hand-fired plants.

10. From a geographical classification of the principal considerations of coal selection, clinkering and its related factor of maintenance stands at the head of the list.

11. There is already established a preference for coal of the high fusing point within limits of choice imposed by the typical characteristics of the coals available in each section. It is the tendency to overreach in the requirements of high fusing coals out of fear of clinker formation more than is necessary for economical operation that tends to create a premium for such a coal.

12. In respect to ash and heat value within any one market division, there is no set-up in preference, and the preference appears to be established more or less in accordance with the delivered price of varying coals.

13. With reference to volatile, with the exception of the Middle Atlantic Division of the market, the economic limits determined the volatile within a very narrow range and practically the only effect of volatile is felt in the design of new plants. There is some importance set upon volatile where strict smoke ordinances are in effect.

14. Fusing point of ash appears to be the only measurable physical characteristic that has a relationship to operation.

15. The question of coking or free burning affecting the choice of coals is not brought out in the reporting in the questionnaires, and appears to show a lack of proper consideration for that question. This is the one physical characteristic of coal for which there is no established method of

comparative measurements, which is very much needed, and I believe some experimental effort is being made to accomplish it at several laboratories at the present time.

16. The reports have a tendency to show that there is nothing inherently inconsistent in the present market divisions resulting largely from the cost of transportation, although it might be possible to classify a more nearly ideal requirement.

17. There is not established from the questionnaires any dividing line between the economy point in the use of coal on a solid bed as compared with pulverizer plants.

Pertaining to Method of Selection of Coals

1. Coal selection by steam plants is still almost universally conducted upon a trial-and-error basis, exceptionally few of the questionnaires revealing any clear-cut relationship between any specific coal characteristics and performance requirements of the plant.

2. Reports of coal selection and fusing temperature of the coals reported clearly indicate in a number of cases a lack of knowledge on the part of the consuming plants.

3. The questionnaires in many cases did not give the proper data on equipment, on operating data, or on the coal analysis and showed a lack of proper control.

4. With reference to the classification of coal for stationary steam boilers from a use standpoint, we do not feel it is possible to establish any hard and fast rules on account of the infinite variety of equipment, design and operating that are found in existing plants. On the other hand, there is a definite requirement for the scientific classification of coal to be adapted by consumers to stationary steam installations.

Conclusion

The results of this survey of the questionnaires show that coal selection is still largely governed in individual cases by trial-and-error methods, instead of by analytical study of the plant to determine exactly *why* coal of a certain kind produces certain results in a given plant.

Progress in simplification of coal selection must depend upon more widespread and more critical knowledge of actual plant requirements and their relation to the character of the coal economically available in each region. Coal buyers undoubtedly have turned hopefully to the laboratory for what it could disclose as to measurable coal characteristics, only to be disappointed by the difficulty of interpreting what the laboratory tells into practical coal-selection methods. The reason undoubtedly lies in the fact that what should have been a parallel movement to analyze critically the *use* of coal in steam plants has lagged behind.

In view of the difficulties of establishing a more satisfactory basis for the classification of the plants themselves, or in other words, for the classification of the use, it may be necessary to continue to develop from boiler tests the characteristics of the coal best suited for the individual plant and the application of a scientific classification of coals to determine in conjunction with the results of the boiler tests which coals available are suitable and which are not.

The questionnaire bears out the fact that more consideration should be given to the character of the coals most economical for delivery; as, for instance, in new installations; to the volatile to establish the proper combustion space and smoke regulation requirements; to the British thermal units for overload requirements; to the amount of ash for cost of handling; to the coking and caking of the coal and fusing point of ash characteristics for the proper type of combustion equipment; to the moisture and friability for pulverizing-plant selection.

DISCUSSION

(*Thomas G. Fear presiding*)

H. T. Coates,* New York, N. Y. (written discussion).—When considering the purchase of coal for definite use, following out the idea expressed on page 133 one would naturally start with the type of burning equipment; that is, hand, overfeed, underfeed stoker, etc. Then if anticlinkering was the predominant factor, would it not be possible to pick out a coal geographical section, certain seams of a group of mines, which would rank high in this feature, perhaps giving different local sections or different mines something to designate a rating ("good," "fair" and "poor")? Do the same thing for free-burning, antislagging, smokelessness, friability and grindability. List these or arrange them on a map or diagram in such a way that the prospective buyer could find a coal that rated particularly high from the point of anticlinkering and still show in the "fair" section of free-burning, grindability or other characteristics to which he desired to give some consideration. Perhaps a reference chart somewhat along the lines I have sketched herein might be the foundation for such a scheme. In this case a particular mine might be rated 2-8-27, 4-16-30. The purchaser would then know that this coal was not likely to clinker when fired by hand

Table 10.—*Suggested Numbering System for Coal Classification*

	Hand	Overfeed	Underfeed	Chain	Pulverizer
Free-burning	1	7	13	19	25
Anticlinkering	2	8	14	20	26
Antislagging	3	9	15	21	27
Smokelessness	4	10	16	22	28
Grindability	5	11	17	23	29
Friability	6	12	18	24	30

A, B, C rating in good, fair and poor.

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[*] Purchasing Agent and Plant Engineer, Dairymen's League Co-Operative Association, Inc.

or an overfeed stoker, but might give some trouble in other styles of equipment. It would not be likely to give serious slagging in pulverized-coal units and there would not be difficulty with smoke either when hand fired or in an underfeed stoker. Used in other ways than shown by the classification, it might give trouble, and *caveat emptor* would govern.

Certain producing sections, then, might be marked with numbers corresponding to the characteristics of the coal, or mines might be listed with the characterizing numbers. Perhaps a preliminary step in the selection of the coal might be to turn to a map with these numbers marked on different sections to show what list of mines would be the one in which to expect what was wanted. In this case the numbers as shown on Table 10 would represent classifications of the coals for each characteristic and use. The combination of the numbers would give a selective classification for certain purposes governed by the type of equipment and coal characteristic desired.

C. A. Reed,* Pittsburgh, Pa. (written discussion).—The author has made a valuable study of the reaction of the purchasers and operators to the kind of coal they prefer and the preference reasons. It is probably the first collection of data of such an extensive nature and should be a pilot light to further definite lines of selection of coal.

This discussion is the reaction we have gained by direct contact with hundreds of plants in part of the buying sections of the country covered in the article. I have purposely not covered any part of the paper dealing directly with scientific classification.

The trend of purchase of economical coal, which is generally conceded to be slack from most mining districts, seems to be in direct relation to the buying and using division that shows the largest consumption of coal. This same district also shows a marked degree of, shall we say, intelligence in the methods of burning this cheap grade of coal.

A conjecture would lead me to believe that coal is burned in the great majority of these plants under large, economical boilers at high ratings of boiler capacity and at high economical heat transfers, as against some other districts that still stand by the old-style heat wasters of small horsepower with hand-fired grates. There is a plant not far from New York where there are installed a great number of 150-hp. H.R.T. boilers, all hand fired, and the owners really believe they are not in a position to save any money by investing, and following the experience of the majority, in equipment that would give them probably 30 per cent more in efficiency, cut out that direct proportion of coal tonnage, reduce labor at least 50 per cent and in many other ways reduce losses to pay for new equipment in from 2 to 5 years.

Referring to Chief Factors Affecting Choice of Coal in Pulverizer Plants, it has been my privilege to see many plants and talk to purchasers and operators who have had the fortune to have equipment so designed as to allow them to take advantage both of freight rates on coal and the sometimes too frequent price changes in some fields. I mean that for very little added first cost, furnaces have been laid out and equipped and pulverizers have been of capacity to allow them to buy and economically use a wide variety of coals.

In some cases the over-all efficiencies of the plants may vary slightly but by selective buying the cost either per thermal unit or million pounds of steam remains within fair limits. In a few cases the plants are limited to almost one narrow group of coals regardless of the change in delivered price per thermal unit. There are too many plants with poorly designed settings, insufficient volumes or no provision for slag prevention, or that must depend on the most friable coals to carry their loads due to a very small saving in pulverizer first cost. The present selling price of coal is not the basis of this statement, as it is generally known that a great proportion of mine operating companies are selling coal at a loss, either book or cash out of surplus.

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* Combustion Engineer, Pittsburgh Coal Co.

In referring to the section on Factors Affecting Choice of Coal Differ According to Type of Equipment, it is especially noticeable that the overfeed stoker is heavily penalized with clinkering. Could this not be traced back to the fact that these machines are usually found in small plants where the operators have few instruments to help them get proper results, and where proper supervision is probably lacking in many cases? These machines are in the main of natural draft design. To be expected to follow any kind of a fluctuating load in the plant would place the coal under severe service, owing to change in draft in the furnace, change in speed of the stoker and lack of cooling and combustion air at time of low demand, to the other extreme of excess air and forced fires to meet peak demand.

Table 6 has a peculiar set-up in that of all groups the Pittsburgh group is the only one where clinkering is considered of prime importance, so that it is placed first in the headings. If this condition is looked into from the standpoint of past experience, it will probably bring to light that some time back many public utilities and large industrial companies purchased coal property for their own use. Much of this property was of a medium and low-grade coal in so far as ash and sulfur were concerned, of which the latter tends to low fusion coal. Other large companies were and still are financially connected in some manner to producing companies owning some coal of this same general class. Companies not so associated with a definite tie-up know well the difficulties that are encountered when using low-grade coals, so that in order to be free of any possible trouble, they specify against this grade by making clinker the first objection. The greater proportion of coal being mined now is high-fusion, low-sulfur and can be obtained from the large and dependable producers in the field. Because of this demand, which is only natural, many of the small mines producing low-grade coals are closed or are selling to companies that do not operate at high overloads or with high over-all efficiency.

In referring to the conclusion of the author's paper, I think that it should be plainly understood, especially by those who do not come in direct contact with either purchasers or the actual operating men who burn the coal, that in many plants—more especially the great number of users of smaller tonnages, a great deal of trial and error goes on continuously in the plant, since, first, many plants are not given complete scientific control and recording instruments to govern their operation; second, the natural variation in all raw coal greatly affects small stokers or pulverizers; third, it would seem, in discussing this variation with hundreds of operating men, that the off-standard coal in a car or a bunker seems to find its way into the furnace too often just when heavy demand is needed and, fourth, the class of coal delivered to the same plant changes often, either because of stock-pile reserves or current or spot purchases, so that unless the men and the equipment are thoroughly controlled they may run into hours of poor efficiency before that condition is checked and corrected.

In probably the greater number of plants burning coal, other than public utilities, the boiler room is considered a necessary evil and the matter of making it profitable is always keep out of the so-called "front office"; the chief engineer, in a word of his own language, has a devil of a time getting the management to see his problem in an open-minded way.

Since the author's paper is prepared to present the study made by the National Purchasing Agents Association to assist in more wisely selecting coal, I will conclude my discussion by saying that coal producers and sellers have found a bad but necessarily true condition in the country, especially with the smaller user of coal. This is entirely due to the fact that the employer expects his purchasing representative to be a genius and have expert knowledge of everything he buys, coal included, whereas in the larger plants he has and uses expert knowledge from the entire organization when necessary.

T. W. Harris, Jr. (written discussion).—It might be well to add to this subject the fact that a new type of stoker called the "spreader stoker" is receiving considerable consideration, particularly through the midwestern area of the United States burning high-volatile coals. This stoker throws the coal into the combustion chamber; part of the coal is burned in suspension and the remainder on a stationary grate. The smaller sizes of coal and slack can be burned on this stoker and it is claimed by the stoker manufacturers that they can burn relatively low fusing-point coal without experiencing difficulty from clinkering.

It would appear that this type of stoker is limited to smaller boiler installations. Forced draft is necessary, which means that care must be taken that the finer sizes of coal are not carried over into the boiler. There do not seem to be enough data available to definitely establish this type of stoker's limitations based on various coals.

Table 2 shows the size of coal used by plants reporting from each division of the market and it is our understanding that these figures are somewhat obsolete at present in view of the considerable further trend towards the use of slack coal, especially since code prices have been established. Over the past year, an effort is being made, in establishing code prices, to base them upon quality differences available to the consumer, and this has had a marked effect in making both the producer and consumer conscious of these quality variations in the coals themselves and the results obtainable in the many various burning conditions in the consumers' plants.

Classification of Coals of the United States According to Fixed Carbon and B.t.u.*

By W. A. Selvig,† W. H. Ode,‡ Pittsburgh, Pa. and A. C. Fieldner,§
Washington, D. C.

(New York Meeting, February, 1934)

By plotting fixed carbon against British thermal units of coals free from mineral matter, and ranging in rank from anthracite to lignite, it is found that the coals of higher rank, from anthracite to the higher rank of the high-volatile bituminous coals, inclusive, can be separated into groups according to their degree of metamorphism by reference to their fixed carbon. Such charts show also that the coals lower in rank can be best separated into groups with respect to B.t.u., especially when plotted on the moist basis, that is, coal containing its natural bed moisture.

A tentative scheme for coal classification was presented at the October, 1932, meeting of the Technical Committee on Scientific Classification, of the American Standards Association Sectional Committee on Coal Classification. This scheme was sponsored by Subcommittee IV on Tentative Classification of Coals, under the chairmanship of W. T. Thom, Jr. It proposed the grouping of coals according to rank by reference to fixed carbon and B.t.u., on the basis of coal containing its natural bed moisture but free from mineral matter. This scheme proposed classifying coals into four broad classes: (1) anthracitic, (2) bituminous, (3) subbituminous, (4) lignitic. The anthracitic class was divided into three groups designated as meta-anthracite, normal anthracite, and semianthracite. The bituminous class also was divided into three groups, low volatile, medium volatile, and high volatile. The high-volatile group was further subdivided into four subgroups according to moisture content, as follows: (1) less than 5 per cent, (2) 5 to 10 per cent, (3) 10 to 15 per cent, and (4) 15 per cent and more. The subbituminous class was divided into two groups, group 1 including subbituminous coals having calorific values of 10,500 B.t.u. or more, and group 2 including those with less than 10,500 B.t.u. The lignitic class of coal was divided into two groups both below 8,000 B.t.u., the one group consisting of consolidated and the other of unconsolidated coals. This classification scheme is mentioned here as it was made the basis for the grouping of coals for the statistical work described in this paper, with a view to establishing suitable boundaries as to fixed carbon or B.t.u. for classifying the coals of the United States.

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The data given in this paper were assembled at the Pittsburgh Experiment Station of the U. S. Bureau of Mines, in cooperation with Subcommittee V on Boundary Lines for Coal Classification, under the chairmanship of W. A. Selvig. Charts were prepared for the coals of higher rank showing fixed carbon plotted against ash, both on the moist basis and on the dry basis. Likewise, similar charts were prepared for the coals of lower rank showing B.t.u. plotted against ash. The analyses used for plotting were those made by the U. S. Bureau of Mines. Average mine values of fixed carbon, B.t.u., and ash were used, except with anthracitic coals, of which only a limited number of mines had been sampled, in which case individual analyses were plotted. The rank designation of the coals was taken as published by the U. S. Bureau of Mines. Lines of equal B.t.u. and fixed carbon were drawn on the charts to show these values expressed on the basis of mineral-matter-free coal; the latter was taken as 1.1 times the ash. With such a simple correction factor it is easy to draw in lines of equal B.t.u. and equal fixed carbon. These lines offer a convenient method for fixing suitable boundaries between different ranks of coal, provided sufficient analyses are available.

ANTHRACITIC CLASS

This class includes three groups—meta-anthracite, normal anthracite, and semianthracite. It should be noted here that the names of the various classes and groups of coal as used in this paper are only for convenience in discussion. The purpose of this investigation is confined to fixing suitable boundaries between different ranks of coal and is not concerned with nomenclature, which comes under the jurisdiction of a committee on nomenclature of the Sectional Committee on Coal Classification, under the chairmanship of E. A. Holbrook.

Only twelve mine samples of meta-anthracite were available and these were from only two mines formerly operating in Newport and Providence Counties, R. I. The bed moisture of these samples ranged from 4.5 to 23.7 per cent, much of which apparently is held loosely, since high-moisture samples of these coals lose moisture readily when exposed to the air. The dry basis is considered preferable for classifying meta-anthracites. These mine samples were high in ash, ranging from 14.4 to 40.6 per cent on the dry basis. When calculated to the dry mineral-matter-free basis, the meta-anthracites showed fixed carbon of 98 per cent or more.

Analyses of 213 mine samples of normal anthracite from 27 mines in Pennsylvania were plotted; these showed a lower limit of 91.4 per cent and an upper limit of 98 per cent fixed carbon, on the dry basis, free from mineral matter.

Coals designated as semianthracites in U. S. Bureau of Mines publications were plotted from Sullivan County, Pa., and from Arkansas and Virginia. In all, 55 mine samples from 37 different mines were available, of which 12 were from Pennsylvania, 11 from Arkansas, and 32 from Virginia. On the dry mineral-matter-free basis, these mine samples came within the limits of 86 and 92 per cent fixed carbon; the samples from Sullivan County, Pa., ranged from 89.5 to 92.0 per cent; those from Arkansas, from 86 to 90 per cent; and those from Virginia, from 86 to 91 per cent fixed carbon.

Bituminous Class

Bituminous coals usually are divided into (1) the low-volatile group, sometimes called semibituminous, (2) the medium-volatile group and (3) several high-volatile groups. The report of Subcommittee IV, previously mentioned, recommended an upper limit of 22 per cent volatile matter, on the moist basis but free from mineral matter, for the low-volatile group. Assuming an average moisture of 3 per cent for these coals, this is equivalent to 77 per cent fixed carbon on the dry basis and free from mineral matter.

No generally accepted upper limit of volatile matter for medium-volatile coal has been established. The report of Subcommittee IV tentatively proposed an upper limit of 32 per cent volatile matter on the moist mineral-matter-free basis. From the standpoint of the coking industry this limit would be too high.[1] Probably a better upper limit would be 30 per cent volatile matter on the same basis. Assuming an average moisture of 3 per cent for this group of coals, this would be equivalent to 69 per cent fixed carbon on the dry basis free from mineral matter. This value has been accepted tentatively by the authors of this paper.

In general, the bed moisture does not vary widely in the case of normal anthracite, semianthracite, low-volatile and medium-volatile bituminous coals, and the higher rank of high-volatile coal. Since the bed moisture of these coals has no special significance in respect to their utilization, and is fairly uniform, it appears best to classify them on the dry basis. This simplifies the problem of sampling, as no special precautions need be taken to insure that the samples have their true bed moisture; consequently, analyses of wet samples can be used for classification purposes.

On the other hand, the high-volatile bituminous coals of lower rank, the subbituminous coals, and the lignites show a variable and relatively high moisture content, which may be of considerable importance as affecting their use. These groups, therefore, are best classified on the basis of coal containing its natural bed moisture, and the B.t.u. values on the moist basis, free from mineral matter, afford the best means of grouping them by rank.

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[1] H. J. Rose: The Selection of Coals for the Manufacture of Coke. *Trans.* A. I. M. E. (1926) **74,** 600–639.

In plotting analyses of high-volatile bituminous coals, these were divided into the following four groups according to their moisture content, the basis including mineral matter: (1) moisture less than 5 per cent; (2) moisture from 5 to 10 per cent; (3) moisture from 10 to 15 per cent; (4) moisture 15 per cent or more. All available analyses representing coals of the interior and the western United States were plotted. The only high-volatile coals considered from the Appalachian field were those from Ohio, as they represent the coal of lowest rank in this field.

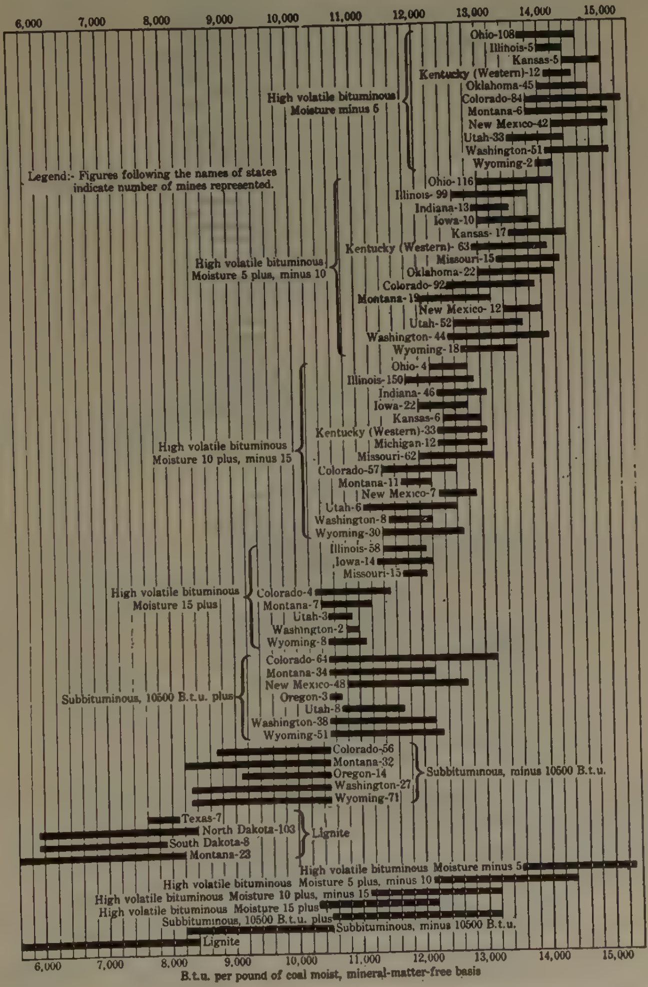

FIG. 1.—B.T.U. LIMITS OF VARIOUS COALS; MOIST, MINERAL-MATTER-FREE BASIS.

The B.t.u. ranges of these high-volatile bituminous coals are shown by states in Figs. 1 and 2. Fig. 1 gives the range in B.t.u. on the moist, mineral-matter-free basis, while Fig. 2 shows the B.t.u. range on the dry basis. Comparison of these two figures, especially the summaries at the bottoms, shows that the overlapping of the various groups is much more pronounced on the dry basis than on the moist basis.

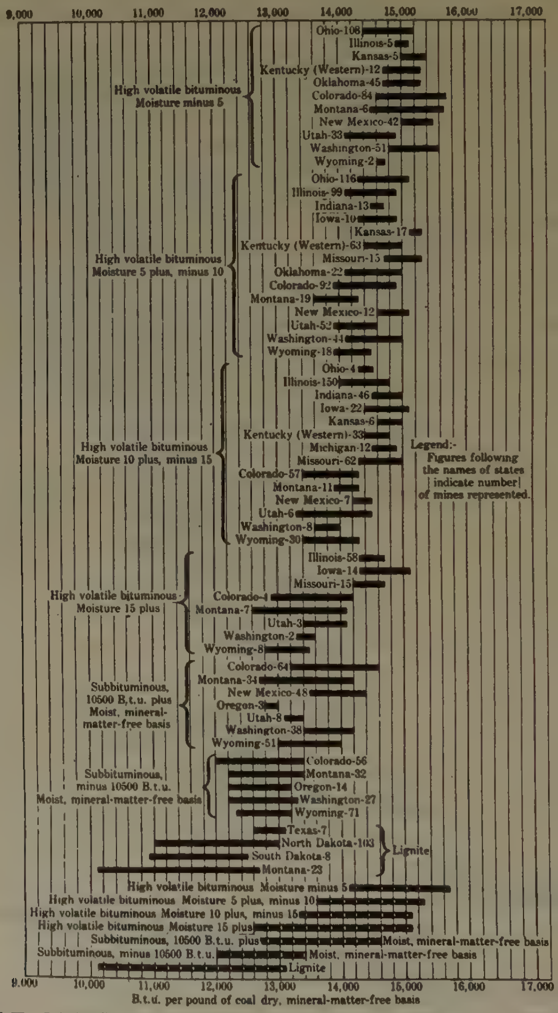

FIG. 2.—B.T.U. LIMITS OF VARIOUS COALS; DRY, MINERAL-MATTER-FREE BASIS.

SUBBITUMINOUS AND LIGNITIC COALS

Similar charts were prepared for subbituminous and lignitic coals. The subbituminous coals were divided for charting according to the scheme proposed by Subcommittee IV previously mentioned; one group included coals with over 10,500 B.t.u. on the moist, mineral-matter-free basis while the other group ranged below 10,500 B.t.u. The ranges of B.t.u. for the subbituminous and the lignitic coals are shown in Figs. 1 and 2. Here again, the overlapping of the B.t.u. ranges for the different groups is most pronounced on the dry basis; hence the moist basis affords the best means for separating these coals into groups.

PROPOSED BOUNDARIES FOR COAL CLASSIFICATION

Table 1 gives the proposed boundaries of fixed carbon and B.t.u. for the various ranks of coal as determined by the plotting of a large

TABLE 1.—*Classification of Coals by Rank*

Class[a]	Group[a]	Limits of Fixed Carbon or B.t.u., Mineral-matter-free Basis	Requisite Physical Properties
I. Anthracitic	1. Meta-anthracite	Dry F.C. 98 per cent or more	
	2. Normal anthracite	Dry F.C. 92 per cent or more and less than 98 per cent	
	3. Semianthracite	Dry F.C., 86 per cent or more and less than 92 per cent	Nonagglutinating[b]
II. Bituminous	1. Low volatile	Dry F.C., 77 per cent or more and less than 86 per cent	
	2. Medium volatile	Dry F.C., 69 per cent or more and less than 77 per cent	
	3. High volatile A	Dry F.C., less than 69 per cent; and moist B.t.u., 14,000 or more	
	4. High volatile B	Moist B.t.u., 13,000 or more and less than 14,000[c]	
	5. High volatile C	Moist B.t.u., 11,000 or more and less than 13,000[c]	Either agglutinating or nonweathering
III. Subbituminous	1. Subbituminous A	Moist B.t.u., 11,000 or more and less than 13,000[c]	Both weathering and nonagglutinating
	2. Subbituminous B	Moist B.t.u., 9,500 or more and less than 11,000[c]	
	3. Subbituminous C	Moist B.t.u., 8,300 or more and less than 9,500[c]	
IV. Lignitic	1. Lignite	Moist B.t.u., less than 8,300	Consolidated
	2. Brown coal	Moist B.t.u., less than 8,300	Unconsolidated

[a] The names given above under "Class" and "Group" are used temporarily, pending recommendations of the Technical Committee on Nomenclature, Sectional Committee on Coal Classification.

[b] If agglutinating, classify in low-volatile group of the bituminous class.

[c] Coals having 69 per cent or more fixed carbon on the dry mineral-matter-free basis to be classified according to fixed carbon, regardless of B.t.u.

number of coals of the United States. According to this scheme the anthracitic coals are divided into three groups, the bituminous coals into five groups, the subbituminous coals into three groups, and the lignitic coals into two groups. The physical properties of agglutination and weathering are used also to differentiate between certain adjacent groups.

Fixed carbon on the dry basis free from mineral matter is used for marking boundaries between meta-anthracite, normal anthracite, semianthracite, low-volatile bituminous, medium-volatile bituminous, and the first group of the high-volatile bituminous coals. B.t.u. on the moist basis free from mineral matter is similarly used for distinguishing between the three groups of high-volatile bituminous coals and for the lower-rank coals. Coals having 69 per cent or more fixed carbon on the dry basis free from mineral matter are grouped according to fixed carbon regardless of their B.t.u. values.

In this scheme it is proposed that coals having calorific values of 14,000 B.t.u. or more on the moist, mineral-matter-free basis, and those having 69 per cent or more fixed carbon on the dry basis free from mineral matter are to be classified according to fixed carbon on the dry basis. Coals having calorific values less than 14,000 B.t.u. on the moist basis free from mineral matter are to be classified according to B.t.u. on the moist basis, provided the fixed carbon on the dry basis is less than 69 per cent.

The basis for selection of the proposed boundaries of fixed carbon for the groups from meta-anthracite to and including the first group of the high-volatile bituminous coals has been stated. Attention is called, however, to the fact that certain coals would be classified as semianthracite or as low-volatile bituminous coal depending on whether or not they agglutinate. If they agglutinate they would be classed as low-volatile bituminous coal and not as semianthracite. Work is now in progress in the laboratories of the U. S. Bureau of Mines at Pittsburgh and in the Fuel Research Laboratories, Canadian Department of Mines, at Ottawa, to decide what laboratory test shall be used to determine what constitutes agglutination for coal classification purposes. Pending agreement on this method of test it is proposed that coals having agglutinating indices of 500 grams or more at a ratio of 15 parts sand to 1 part of coal by the U. S. Bureau of Mines method[2] are to be considered as agglutinating.

Fig. 1 indicates that it would be feasible to divide high-volatile bituminous coals arbitrarily into three groups according to B.t.u. on the moist basis, thus: group A, 14,000 or more; group B, from 13,000 to 14,000; group C, from 11,000 to 13,000. According to this scheme, the high-volatile bituminous coals of the Appalachian field will fall in group A, with the exception of some of the Ohio coals. Most of the Mid-Continent bituminous coals would, according to this plan, come in group B and group C.

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[2] W. A. Selvig, B. B. Beattie and J. B. Clelland: Agglutinating Value Test for Coal. *Proc.* Amer. Soc. Test. Mat. (1933) **33**, II.

Fig. 1 shows a pronounced overlapping in B.t.u. between certain subbituminous coals and the high-volatile bituminous coals. Obviously, therefore, B.t.u. values alone are not sufficient to fix the boundaries between these groups. It is proposed that coals coming within the range of 11,000 to 13,000 B.t.u. on the moist basis free from mineral matter be classified as high-volatile bituminous or as subbituminous depending on their weathering and agglutinating properties. If they are either agglutinating or nonweathering they are to be classified as high-volatile bituminous; if they are both weathering and nonagglutinating they are to be classified as subbituminous. As to what shall constitute weathering, it is proposed that coals showing first cycle weathering or slacking indices of 5 per cent or more, as determined by the U. S. Bureau of Mines test[3] shall be considered weathering or slacking coals. For this test, 500 to 1000 grams of approximately 1 to 1½-in. lumps of coal are air-dried at a temperature of 30 to 35° C. for a period of 24 hr., and then immersed in water for 1 hr. The water is then drained off and the sample again air-dried for 24 hr. The amount of disintegration is determined by sieving on an 8-in. wire-mesh sieve with 0.263-in. square openings. The amount of undersize and oversize is weighed and the percentage of undersize, after deducting a blank sieving test, is taken as the weathering or slacking index. Experimental work is now in progress to determine whether it is necessary to control the humidity of the air-drying oven used in this test.

Table 1 shows that the subbituminous class has been divided into three groups according to B.t.u. on the moist, mineral-matter-free basis; namely, group A, from 11,000 to 13,000; group B, from 9,500 to 11,000; group C, from 8,300 to 9,500. This proposed division has been made arbitrarily to cover the range in B.t.u. for subbituminous coals as shown in Fig. 1.

Fig. 1 shows very little overlapping between the lignites and the subbituminous coals. Accordingly, the boundary would best be placed at 8300 B.t.u. on the moist basis. As stated before, in plotting the various coals shown in Fig. 1 the coals were grouped by rank according to the designations given in Bureau of Mines publications of coal analyses. Undoubtedly some coals were designated as lignites that might better have been put in the subbituminous class, and vice versa. For this reason the proposed boundary of 8300 B.t.u. as shown in Table 1 is subject to revision.

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[3] A. C. Fieldner, W. A. Selvig and W. H. Frederic: Accelerated Laboratory Test for Determination of Slacking Characteristics of Coal. U. S. Bur. Mines *Rept. of Investigations* 3055 (1930).

Under the lignitic class two groups are proposed; namely, (1) lignite, and (2) brown coal. Lignite is the predominant group of this class in this country. So far as known, very little brown coal occurs in North America, though some occurs in Illinois, Texas and Canada.

The authors of this paper realize that the boundaries proposed in Table 1 are subject to revision as more data are accumulated. The scheme proposed, however, does offer a feasible plan for coal classification based on a statistical study of a large number of coals of the United States.

DISCUSSION

(*L. E. Young presiding*)

G. H. CADY,* Urbana, Ill.—With reference to classification on a moist basis, the relative importance of variations in moisture as compared with variations in calorific value should be considered. Since moisture in general increases as calorific value decreases, combining moisture and calorific value might appear simply to accentuate differences and hence simply to spread the range of calorific value for any coal. However, as the moisture increases it is apparent that the moisture factor in the moist B.t.u. values becomes of increasing importance, so that classification of the high-moisture coals comes to be essentially a classification on the basis of variations in moisture. It would seem to be better either to use moisture entirely or not to use it at all. The objection to the use of moisture lies in the irregularity in the moisture content of coals having more than 10 per cent moisture, so that precision of comparison and evaluation is impossible. The question resolves itself into one of choice between similarities when comparison is made on the dry B.t.u. basis and of differences when comparison is on a moist basis. In the paper under consideration the emphasis is on the differences due to variations in moisture.

E. Stansfield,† Edmonton, Alta.—The classification was essentially that already in use by the Bureau of Mines and the authors attempted to establish definite boundary lines in agreement with the existing classification. This was probably inevitable, although it might have been better if the boundary lines could have been placed to best advantage regardless of established custom. Fig. 1, however, showed coals called bituminous down to 10,300 B.t.u., and coals called subbituminous as high as 13,200 B.t.u. A similar overlap may exist between anthracite and bituminous coal. Anthracites and bituminous coals are used for different purposes and the placing of a coal in the one class or the other does not indicate superiority or inferiority. The name subbituminous, on the other hand, implies an inferior fuel, which may be incorrect. A new name is needed. As to the relative merits of the dry and moist bases for classification by calorific value, the emphasis should be placed on the fuel as it goes to the consumer, not on the fuel as dried in the chemist's oven.

R. P. HUDSON,‡ Wayland, Ky. (written discussion).—The authors have suggested what seems to be a reasonable method of classifying coals. The degree of metamorphism, as determined by means of fixed carbon and B.t.u. values, is no doubt a more or less reliable index of the rank of a coal. I agree that any scheme of classification for the lower rank coals should be based on the analysis of the coal containing its natural bed moisture, but free from mineral matter. I also agree that the mineral matter may be estimated with sufficient accuracy by multiplying the ash content by 1.1.

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† Chief Chemical Engineer, Research Council of Alberta.

‡ Chemist, Elk Horn Coal Corporation.

In speaking of coals of the bituminous class, the authors state that Subcommittee 4, of the American Standards Association Sectional Committee on Coal Classification, recommended an upper limit of volatile matter of 22 per cent, on the moist basis but free from mineral matter, for the low-volatile group. I agree with this recommendation. It is true that there is a considerable acreage of coal in West Virginia, in Greenbrier County and elsewhere, which runs from 25 to 30 per cent volatile matter and which is often classed as low-volatile coal. However, a boundary line must be drawn somewhere and the upper limit of true low-volatile coal is usually considered to be 23 per cent volatile matter.

The authors note that no generally accepted limit of volatile matter for medium volatile coal has ever been established. The so-called natural coking coals would fall into this category. In the absence of any generally accepted figure, the authors tentatively accept 30 per cent volatile matter, on the moist mineral-matter-free basis, as the upper limit of volatile matter for coals in this classification. In this connection, it might be well to state that Connellsville coal, long considered an ideal coking coal, will average approximately 30 per cent volatile matter, on the as-received basis.

From information scattered through metallurgical literature, I conclude that the most economical coal to use in the byproduct oven, in making blast-furnace coke, is one that requires about 1.4 tons to produce a ton of coke. A brief calculation will show that such a coal, with 3 per cent moisture, will contain, theoretically, 25.6 per cent volatile matter, on the as-received basis. Assuming that the coke must not contain more than 10 per cent ash, the maximum ash content of the coal will be, therefore, 7.2 per cent and the mineral matter content of the coal will be 7.9 per cent. The complete proximate analysis of the coal, as received, would be as follows: moisture, 3.0 per cent; volatile matter, 25.6; fixed carbon, 64.2; ash, 7.2. The mineral-matter-free volatile matter is evidently 27.8 per cent, on the assumption of the above figures.

The authors realize that analytical data alone are insufficient to fix boundary lines between certain adjacent groups of coal. Therefore the physical properties of agglutination and weathering have been used. The desirability of data concerning agglutinating-value determinations is becoming more generally realized as we attempt to classify and evaluate coals. Work along this line should be encouraged.

Probably the chief value of the authors' paper lies in the fact that their proposed method of classification is based on a statistical study comprising analytical data derived from a large number of coals of various ranks.

Southern High-volatile Coals for Gas and Metallurgical Uses

By Howard N. Eavenson,* Pittsburgh, Pa.

(Hazleton Meeting, October, 1932)

Prior to 1907 nearly all coke was made in beehive ovens, and most of the gas produced was made in the old-style gas retorts, and while there were a few coke plants in southern West Virginia, southwest Virginia and Kentucky operating on high-volatile coals the proportion of the total product made by them was small and usually was used locally. The greater part of the coke made, and the coal gas produced, came from the Pittsburgh seam of western Pennsylvania and northern West Virginia, which was then regarded as the standard coking and gas coal of the country, and the best coal available anywhere for these purposes.

With the advent of the byproduct oven and the building of coke plants at the furnaces, the restriction of the use of one grade of coal for making coke was removed and the use of mixtures of low-volatile and high-volatile coals began. The change in the locations of the coke plants removed the necessity of using Connellsville coal and also made sources of gas available to the public utilities other than of their own manufacture. Most of the first oven plants were built at points distant from the Connellsville region and this afforded an opportunity for the use of coals from the southern Appalachian fields, which were then at approximately equal distances from the ovens.

Since that time development has been made on a number of seams of coal of great purity in southern West Virginia, southwestern Virginia and eastern Kentucky, which have been largely used for gas, cokemaking and metallurgical purposes where low amounts of ash, sulfur and phosphorus are determining factors. These coals are much lower in the objectionable elements than most of the coal shipped from the Pittsburgh seam, or than any northern coal, and when blended with low-volatile coal produce coke in byproduct ovens that is not excelled for any purpose by any coal mined in this country, and is equaled very seldom. Coal from several of these seams is shipped in great quantities without beneficiation and is lower in ash and sulfur than any but the best washed product from the Pittsburgh seam in the north.

The writer has made a study of these coals to ascertain their relative qualities and the areas and reserves of coal available in each of them. No attention has been paid to the coals of Alabama, which do not compete with any of these coals, and no coal of equivalent quality is found in

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Tennessee. Coals of northern West Virginia and western Pennsylvania of quality suitable for metallurgical purposes have not been considered in this study. The study has been made from an engineering viewpoint and not from a geological one; many of the data are contained in the geological surveys of the various states, but many are from personal notes, and none have been assembled in this form before.

LOCATION AND GEOLOGICAL POSITION OF COALS

The area in which these coals are located is in a quadrangle approximately 165 miles long by 48 miles wide, of which the center is nearly where the three states of Kentucky, Virginia and West Virginia meet on Tug River. This area of about 7900 square miles includes parts of Kanawha, Fayette, Raleigh, Boone, Logan, Wyoming and Mingo counties of West Virginia; Buchanan, Dickinson, Wise and Lee counties of Virginia, and Martin, Floyd, Pike, Knott, Letcher, Leslie and Harlan counties of Kentucky. The area is shown on Fig. 1.

During recent years much work has been done in this area in correlating the various seams, a difficult matter in this type of country, with rocks of nearly the same character and with few strata which can be definitely identified over even short distances. The relations of the seams are now well known, with possibly a few exceptions.

These coals all occur in the Pennsylvanian series of the Carbon-

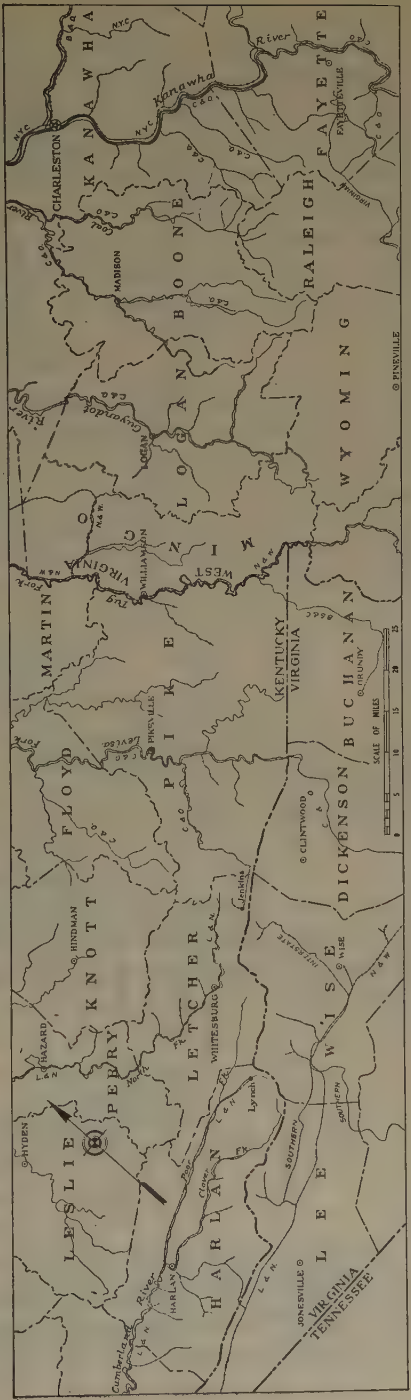


FIG. 1.—HIGH-VOLATILE METALLURGICAL COAL FIELDS IN SOUTHERN WEST VIRGINIA, KENTUCKY AND VIRGINIA.

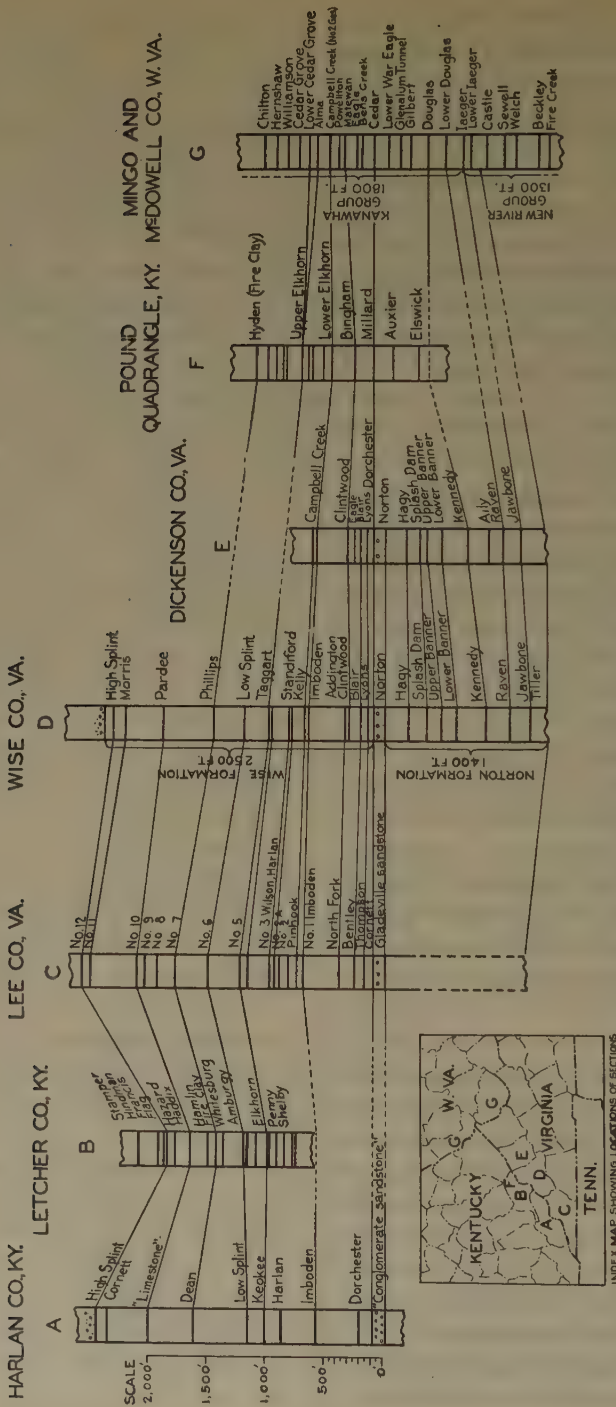


FIG. 2.—SECTIONS SHOWING CORRELATIONS WITH ADJOINING AREAS.

iferous system. These rocks are all of the Upper Pottsville series, and are known as the Kanawha series in West Virginia and as the Norton, Gladeville and Wise formations in Virginia and eastern Kentucky. Along the Kanawha River the strata in this series are about 1200 ft. thick; they increase in thickness toward the south, and along Tug River, near Williamson, the thickness is about 1800 ft. and in the western part of Wise County, Virginia, and in Harlan County, Kentucky, the total thickness is about 3800 ft. A general section is shown on Fig. 2. All of these strata are older than the Pottsville measures in Pennsylvania, and were deposited before any of the coal seams in the western part of the state.

Coal Seams

The number of coal seams in this series varies in different places, from 8 to 24, most of which are of workable thickness in some part of the area but none is of such thickness over the entire area. Sometimes five of these seams are of present commercial value in the same area, but there are rarely more than three, frequently only two, and sometimes only one in this condition. The thickness of the seams varies materially within comparatively short distances and the areas of coal that can be mined commercially now are not large, as compared with the uniform conditions of the Pittsburgh seam.

Nature of Coals Considered

In making this study only the very highest quality of coal for metallurgical purposes was considered, and only seams having face sample analyses of less than 6 per cent ash and 1 per cent sulfur have been considered. These limits are low and exclude many seams which, if cleaned mechanically, will yield coal that will equal this ash specification and will be better than the one for sulfur. The further qualification was made that the coal could be produced, in commercial competition, under market conditions existing in the summer of 1931. Under these conditions the following seams qualify; they are listed in their geological order:

Chilton
Upper Elkhorn—Alma
Roda—"C"—Taggart
Harlan
No. 2 Gas
Imboden
Lower Elkhorn
Powellton
Eagle

The correlations of the various seams in various localities are shown in Fig. 2.

In addition to these there are one or two captive plants operating local seams which produce the best quality of coal, but the total tonnage of this is relatively unimportant. In order to meet the commercial qualification no seam, or portion of a seam, less than 42 in. thick has been considered.

Description of Seams

Eagle Seam.—The portions of the Eagle seam, Fig. 3, within our limitations are found in Raleigh, Fayette, Kanawha, Boone and Logan counties; the small areas in McDowell and Wyoming counties are too high in sulfur. The valuable areas are usually small and scattered, being near the tops of the hills; where the coal is near or below water level, with a few exceptions, the seam is too thin or too badly split by partings to be minable. The seam varies in thickness up to 15 ft., sometimes clean but usually with a streak of bone coal and a slate parting, varying in thickness up to 12 in. Top conditions are never good, and are worse where the coal is the thickest, but usually roof coal can be left up in these places. Bottom is usually fair and grades light. The coal is soft in structure and easily mined.

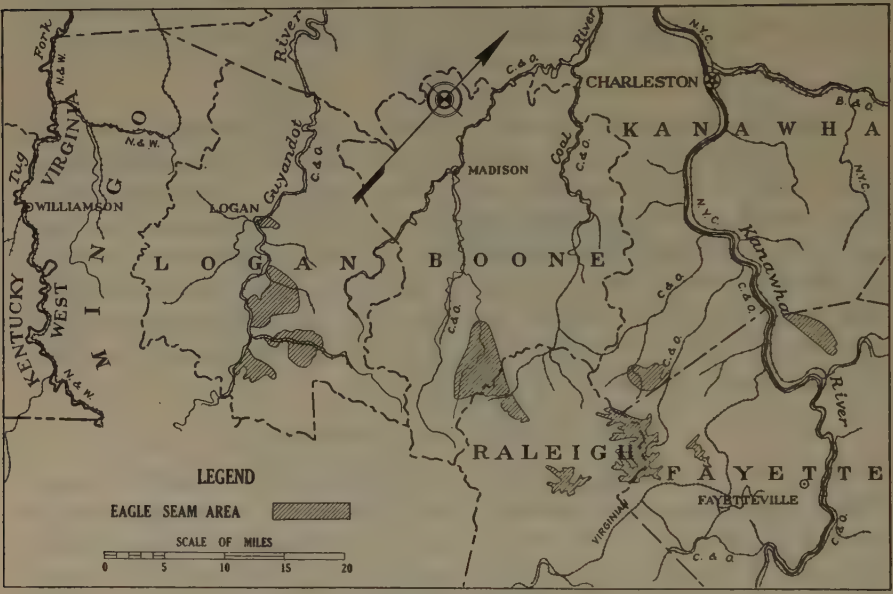


Fig. 3.—Minable area in Eagle seam.

The greatest area of the seam is in Boone County, with Logan County next. The greatest average thickness is in Logan County, with Boone County next. The condition of the seam where it is over 3½ ft. thick, and within our limitations, is shown in Table 1.

Powellton Seam.—The Powellton seam (Fig. 4) is found, within our limitations, in a fair-sized area in Fayette County but with small adjacent areas in Raleigh and Kanawha counties. It has a maximum thickness of 86 in. and usually has a slate parting and sometimes a streak of bony coal. Top conditions are usually fair to good; bottom is soft. Grades

TABLE 1.—*Condition of Seams*

County	Number of Measurements	Thickness, In.								
		Clean Coal			Impurities			Total Seam		
		Max.	Min.	Aver.	Max.	Min.	Aver.	Max.	Min.	Aver.
EAGLE SEAM (where over 3½ ft. thick)										
Raleigh, W. Va.	25	90	44	58.3	12	0	3.2	91	45	61.5
Fayette, W. Va.	43	85	42	51.1	10	0	4.1	86	45	62.2
Kanawha, W. Va.	18	60	46	52.2	11	0	6.7	70	46	58.9
Boone, W. Va.	5	173	42	83.0	12	2	4.4	185	45	87.4
Logan, W. Va.	33	71	44	55.7	10	0	2.0	71	45	57.7
Total average	124	173	42	57.7	12	0	3.7	185	45	61.4
POWELLTON SEAM										
Fayette, W. Va. Raleigh, W. Va. Kanawha, W. Va.	70	77	43	60.4	12	0	4.6	86	43	65
NO. 2 GAS SEAM										
Pike, Ky.	52	80	42	55.8	12	0	5.6	85	42	61.4
Martin, Ky.	3	57	48	54.0	3	0	2.0	59	48	56.0
Mingo, W. Va.	6	72	48	59.7	3.5	0	0.9	72	48	60.6
Logan, W. Va.	16	76	44	58.1	8	0	1.1	76	44	59.2
Fayette, W. Va.	34	77	42	55.7	8	0	1.8	79	42.5	57.5
Total average	111	80	42	56.2	12	0	3.5	85	42	59.8
IMBODEN SEAM										
Lee, Va.	7	81	46	63.6	6	0	2.5	86	46	66.1
Wise, Va.	23	116	44	66.7	8.5	0	2.2	116	44	68.9
Total average	30	116	44	66.0	8.5	0	2.2	116	44	68.2
LOWER ELKHORN SEAM										
Pike, Ky.	50	82	42	52.4	12	0	0.7	94	42	53.1
HARLAN SEAM										
Harlan, Ky.	150	105	42	52.7	12	0	1.9	112	42	54.6
UPPER ELKHORN SEAM										
Letcher, Ky.	106	127.5	42	64.2	11	0	2.6	133.5	42	67.8
Pike, Ky.	98	98	42	59.5	12	0	2.5	109	42	62.0
Floyd, Ky.	30	64	42	47.1	10	0	0.5	64	42	47.6
Knott, Ky.	10	98	42.5	54.3	7	0	3.0	105	42.5	57.3
Total average	244	127.5	42	57.8	12	0	2.2	133.5	42	60.0
RODA SEAM										
Harlan, Ky.	75	72	42	53.4	4.5	0.5	0.2	72	42	53.6
Letcher, Ky.	16	61	42	50.4	12	0	1.8	61.5	42	52.2
Lee, Va.	13	62.5	43	50.8	8	0	1.3	63.5	43.5	51.2
Wise, Va.	35	76.5	42	55.7	12	0	2.9	80	42	58.6
Total average	139	76.5	42	53.4	12	0	1.2	80	42	54.6
CHILTON SEAM										
Logan, W. Va.	56	100	48	64.5	12	0	3.8	105	48	68.3

are light. The coal is soft and bright. The condition of the seam, within our limitations, is shown in Table 1.

No. 2 *Gas Seam.*—The No. 2 gas seam (Fig. 5) is found, within our limitations, only in Fayette, Logan and Mingo counties in West Virginia, although it is present in workable thickness but higher in ash and sulfur in several adjacent counties. In Mingo County it is called the Pond Creek seam, and it is developed extensively, and within our limitations in neighboring Pike County, Kentucky, whence the name Pond Creek comes. There is a small area in Martin County, Kentucky. Further south and west it is variously called the Lower Elkhorn and Imboden seam, but as coal from these seams comes from widely separated areas and is sold under different trade names, it will be treated separately.

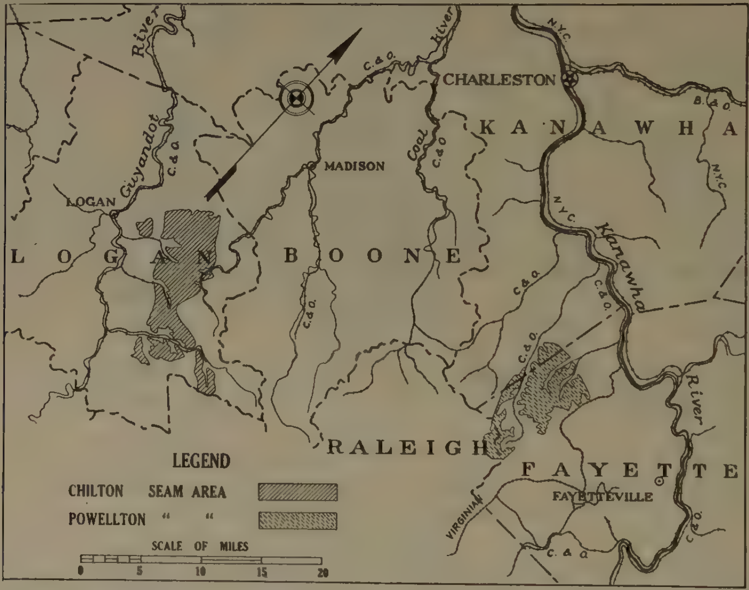


Fig. 4.—Minable areas in Chilton and Powellton seams.

By far the largest area is in Pike County, Kentucky; the amounts in the three West Virginia counties are much smaller. In Pike, Martin and Mingo counties the seam always carries a thick streak of laminated, high-ash coal; in the other counties a slate parting usually occurs. Top conditions are sometimes excellent and always fair; bottom is hard and grades are light. The coal is somewhat harder than the Eagle and Powellton seams. The condition of the seam, within our limitations, is shown in Table 1.

Imboden Seam.—In the western part of Wise County, Virginia, the Imboden seam (Fig. 6) has been mined for many years and the coal used in beehive ovens. This seam is now correlated with the No. 2 Gas seam of West Virginia and the Pond Creek seam of Pike County, Kentucky. The Imboden seam, where worked, has a maximum thickness of nearly 10 ft.

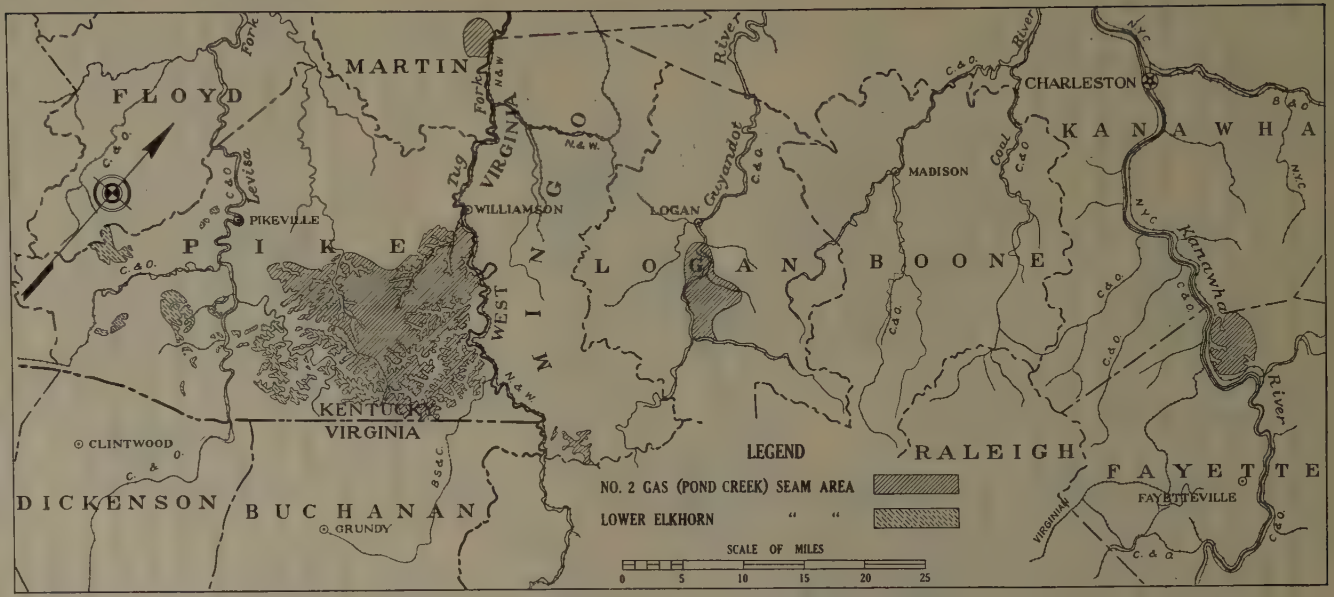


Fig. 5.—Minable areas in No. 2 Gas (Pond Creek) and Lower Elkhorn seams.

and an average thickness of between 5 and 6 ft. of coal with a parting 6 to 8 in. thick. The bed is featured by zones of "curly" coal. Grades are light, and top and bottom conditions are usually good. In adjoining Lee County, Virginia, the seam has been worked in the Pocket, where it ranges from 2½ to 6½ ft. thick. Occasionally the seam is clean but frequently the parting thickens up and makes the seam unworkable. The condition of the seam, within our limitations, is shown in Table 1.

Lower Elkhorn Seam.—In Pike County, Kentucky, west of Levisa Fork of Big Sandy River and mainly on the waters of Marrowbone Creek and the vicinity, the lower of the two seams being worked is called the Lower Elkhorn (Fig . 5) and is correlated with the Imboden seam of Virginia, the No. 2 Gas seam of West Virginia and the Pond Creek seam of eastern Pike County, Kentucky. Where worked the seam is usually clean and of an exceedingly high quality, but is nearly always overlain by a heavy layer, from 6 to 12 in., of layers of coal and slate which break easily and are difficult to separate from the coal. Where this laminated coal is absent a thin slate parting is usually found close to the top of the seam. Grades are light and top and bottom conditions are usually good. This seam may be one of those worked in Floyd County, the identities of which are not exactly known. The condition of the seam, within our limitations, is given in Table 1.

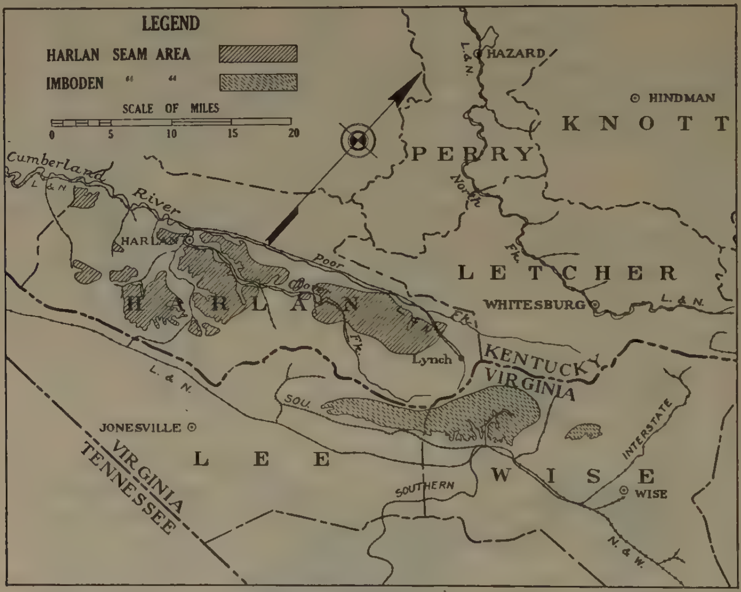


FIG. 6.—MINABLE AREAS IN HARLAN AND IMBODEN SEAMS.

Harlan Seam.—The area of Harlan seam, within our limitations, is found entirely in Harlan County, Kentucky (Fig. 6). The seam is

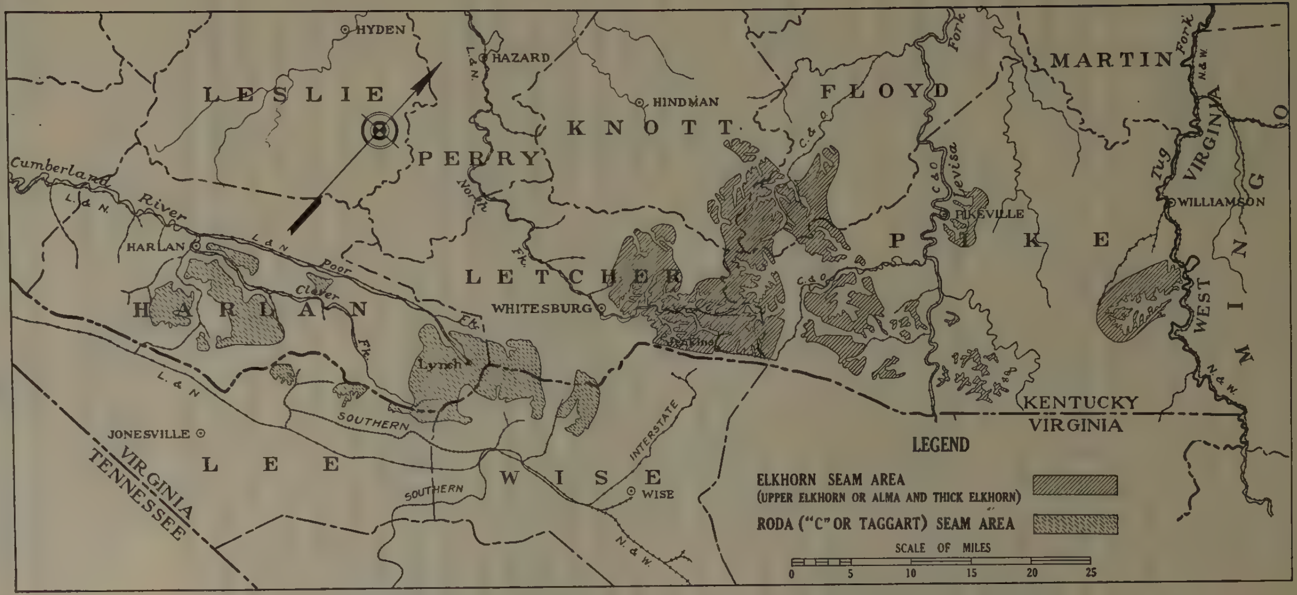


FIG. 7.—MINABLE AREAS IN ELKHORN AND RODA ("C" OR TAGGART) SEAMS.

thick enough to mine commercially in adjoining Lee County, Virginia, but the sulfur content is too high. Where mined it varies in thickness from 3 to over 4 ft., and is almost always clean, with a good bottom and fair to poor top. In the eastern part of the county the seam is much thicker, with better top, but usually has one or more thin partings and a sulfur content over 1 per cent. Grades are usually light. Within our limitations, the condition of the seam is shown in Table 1.

Upper or Thick Elkhorn Seam.—This seam, within our limitations, is found in Pike, Letcher, Floyd and Knott counties, Kentucky (Fig. 7). It is the same as the Alma seam of West Virginia, which in Mingo County is of excellent quality but too thin for present commercial mining. Its main development is in Letcher County, where it is being extensively mined. Where mined its thickness varies from 3½ to 11 ft.; the thinner areas are nearly always clean, and the thicker ones usually have an irregular slate parting varying in thickness up to 30 in., but sometimes in places there are several bone partings with a streak of laminated coal. The clean coal of this seam is one of the purest coals in the world. Top and bottom conditions are good to fair and grades are moderate. Within our limitations the conditions of the seam are shown in Table 1.

Roda, Taggart, Keokee and "C" Seams.—This seam (Fig. 7), as it is variously called in Wise County, Virginia, and Harlan County, Kentucky, supplies most of the highest grade of metallurgical coal produced in the southern fields. Its total area is south of the Pine Mountain fault, but it has now been correlated definitely with the Upper Elkhorn seam. Where mined its thickness varies from 3 to nearly 7 ft. and it is almost invariably clean, excepting in the northeast part of the area where a slate parting is encountered which developes into a split in the seam. Grades vary from light to moderate, bottom is fair and top varies from good where the seam is thinner to fair or bad where it is thicker. The condition of the seam, within our limitations, is shown in Table 1.

Chilton Seam.—The Chilton seam (Fig. 4), within our limitations, is found in Logan County, West Virginia. Where mined it varies from 4 to nearly 9 ft. in thickness. Usually it has one or two streaks of bone, or high-ash coal, a good bottom and fair to poor top. Grades are usually light, but become heavy in the neighborhood of the Warfield anticline. The coal is soft and has an unusually high ash fusing temperature for a coking coal. The condition of the seam, within our limitations, is shown in Table 1.

QUALITIES OF SEAMS

The coal from these seams usually increases in hardness toward the west, both generally and locally. The West Virginia seams are usually softer than those in Virginia and Kentucky, where it is the usual practice

TABLE 2.—*Proximate Analyses, Southern High-volatile Metallurgical Coals (Mine Samples, As-received Basis)*

SEAM	COUNTY	STATE	NUMBER OF SAMPLES	MOISTURE	VOLATILE MATTER	FIXED CARBON	ASH	SULPHUR	B.T.U.	ASH FUSION TEMPERATURE °F	VOLATILE MATTER		ASH		SULPHUR		ASH FUSION TEMPERATURE °F	
											MAXIMUM	MINIMUM	MAXIMUM	MINIMUM	MAXIMUM	MINIMUM	MAXIMUM	MINIMUM
RODA (TAGGART)	Harlan	Kentucky	27	2.23	34.29	60.50	2.98	0.65	13,960	2,358	38.0	36.5	4.9	1.3	0.8	0.4	2,510	2,200
	Letcher (a)	"	1	1.40	36.80	56.40	5.40	0.70	14,370	-	-	-	-	-	-	-	-	-
	Lee	Virginia	24	3.61	36.47	57.30	2.62	0.50	14,040	2,255	38.4	34.1	4.6	1.7	0.8	0.3	2,680	2,040
	Wise	"	9	1.89	34.13	61.77	2.21	0.57	14,660	2,329	35.5	31.3	3.0	1.9	0.7	0.5	2,370	2,196
	Average	Ky. & Virginia	61	2.71	35.16	59.36	2.77	0.58	14,140	-	-	-	-	-	-	-	-	-
ELKHORN	Floyd	Kentucky	6	2.25	36.03	56.14	5.58	0.97	13,620	-	37.1	34.8	9.1	3.6	1.3	0.7	-	-
	Letcher	"	33	2.93	36.50	57.07	3.50	0.66	14,130	2,508	38.3	33.8	7.5	2.4	1.7	0.5	2,730	2,300
	Pike (b)	"	18	2.20	31.83	60.84	5.13	0.68	14,250	2,260	35.7	31.9	9.4	1.1	1.4	0.5	-	-
	Average	"	57	2.63	34.98	58.16	4.23	0.70	14,080	-	-	-	-	-	-	-	-	-
POWELLTON	Fayette	West Virginia	195	1.82	30.64	63.30	4.24	0.77	14,370	-	34.4	28.2	9.0	3.2	1.4	0.6	-	-
	Kanawha	"	1	0.50	43.60	53.60	2.30	0.60	14,330	-	-	-	-	-	-	-	-	-
	Average	"	196	1.81	30.70	63.26	4.23	0.77	14,370	-	-	-	-	-	-	-	-	-
EAGLE	Boone	West Virginia	4	1.32	33.91	59.71	5.06	0.60	14,120	-	39.9	28.4	7.3	2.8	0.6	0.6	-	-
	Fayette	"	32	2.69	29.71	62.62	4.98	0.85	14,360	2,788	44.9	24.5	7.0	3.3	1.6	0.5	3,010	2,390
	Kanawha	"	7	1.56	31.23	63.07	4.14	0.73	14,650	-	35.0	28.9	5.7	3.1	0.8	0.7	-	-
	Logan (c)	"	4	-	31.94	63.80	4.26	0.64	-	-	32.7	30.2	5.1	4.1	0.7	0.6	-	-
	Raleigh	"	10	1.99	28.74	64.64	4.63	0.66	-	-	31.7	26.0	7.7	1.5	0.9	0.5	-	-
	Average	"	57	2.31	30.02	62.90	4.77	0.77	14,420	-	-	-	-	-	-	-	-	-
NO. 2 GAS (POND CREEK)	Fayette	West Virginia	16	2.09	34.99	57.74	5.18	0.96	14,230	2,628	36.7	32.5	8.1	1.7	1.4	0.7	3,010	2,270
	Logan	"	3	1.77	35.13	58.40	4.70	0.70	14,480	2,849	35.7	34.0	4.9	4.5	0.7	0.7	2,960	2,730
	Mingo	"	4	1.87	29.48	62.80	5.85	0.87	14,060	2,540	34.1	27.4	7.8	4.8	1.7	0.5	-	-
	Martin	Kentucky	1	1.60	43.72	49.98	4.70	0.86	14,670	-	-	-	-	-	-	-	-	-
	Pike	"	6	1.50	34.06	58.67	5.77	0.77	14,080	2,940	37.5	31.6	7.1	4.8	1.1	0.6	-	-
	Average	West Va. & Ky.	30	1.90	34.37	58.41	5.32	0.88	14,230	-	-	-	-	-	-	-	-	-
LOWER ELKHORN (d)	Pike	Kentucky	16	1.86	32.17	62.06	3.91	0.60	14,600	-	35.5	27.7	7.2	1.1	1.0	0.4	-	-
HARLAN	Harlan	Kentucky	62	3.02	36.97	56.25	3.76	0.85	14,030	2,643	39.3	35.0	7.3	2.0	1.4	0.5	3,010	2,240
CHILTON	Logan	West Virginia	20	1.89	35.70	57.04	5.37	0.73	14,140	2,727	40.4	31.5	9.3	3.5	1.3	0.6	2,950	2,570
IMBODEN	Wise	Virginia	15	2.28	34.39	57.74	5.59	0.74	13,958	2,740	35.7	33.0	10.4	1.2	0.9	0.6	2,980	2,390

NOTES: Data from Bureau of Mines, State Geological Survey Reports, and Private Records.
(a) Outcrop Sample.
(b) Upper Elkhorn and Thick Elkhorn.
(c) Dried at 105° C.
(d) Probably same as No. 2 Gas.

to screen out the lump coal and sell it as domestic fuel. None of these seams produce coal as hard as that from the splint seams of this area.

Most of these seams have been extensively sampled and analyzed by the U. S. Bureau of Mines and the various state geological surveys, although most of this work, naturally, has been in the developed areas. All of the data available from these sources, as well as many samples made under the writer's direction, have been included in Table 2, showing proximate analyses, heating values and ash fusing temperature. The exceptional purity of these coals in respect to ash and sulfur should be noted, and most of these coals are sold under guarantee of ash and sulfur

TABLE 3.—*By-product Analyses, Southern High-volatile Metallurgical Coals*

SEAM	LOCATION	NUMBER OF SAMPLES	VOLATILE MATTER	FIXED CARBON	ASH	SULPHUR	PHOSPHORUS	GAS YIELD CU. FT. (0°C-30"M)	B.T.U. PER CU. FT. OF GAS	COKE YIELD %	AMMONIUM SULPHATE LBS.	BENZOL LBS.	TAR GALLONS
RODA (TAGGART)	Harlan Co., Ky.	Aver. 12	36.24	60.92	2.84	0.56	.004	10,349	586	69.9	25.4	29.4	7.5
		Max.	38.68	63.27	4.55	0.69	.008	10,868	626	71.0	26.8	33.6	9.2
		Min.	34.68	57.45	1.50	0.44	.001	9,750	538	68.3	20.4	16.4	3.8
	Wise County, Va.	Aver. 8	34.74	62.52	2.74	0.55	.003	10,612	594	71.6	24.2	30.0	7.9
		Max.	36.31	63.33	4.77	0.63	.005	10,808	606	73.1	26.2	32.0	9.6
		Min.	31.94	60.99	2.06	0.45	.002	10,300	582	70.8	22.6	24.4	6.4
ELKHORN	Floyd Co., Ky.	Aver. 13	38.66	57.99	3.35	0.75	.009	10,411	596	68.4	29.3	32.6	8.4
		Max.	40.33	60.48	4.55	0.90	.029	10,578	615	69.7	32.6	35.5	10.2
		Min.	37.06	55.80	1.54	0.60	.002	10,112	579	66.9	26.2	31.3	7.6
	Letcher Co., Ky.	Aver. 7	36.93	60.12	2.95	0.59	.002	10,226	592	69.7	27.5	30.7	8.4
		Max.	37.60	60.94	3.36	0.71	.004	10,294	604	70.6	30.4	32.7	9.2
		Min.	36.14	59.52	2.60	0.51	.001	10,138	588	69.4	26.0	28.4	7.0
	Pike County, Ky.	Aver. 5	33.99	62.22	3.79	0.58	.004	10,769	594	70.8	24.5	24.9	7.8
		Max.	37.06	64.75	6.64	0.70	.007	11,690	631	73.4	26.6	28.4	8.5
		Min.	31.50	59.58	1.06	0.55	.002	9,692	578	70.3	23.6	19.9	6.0
POWELLTON	Fayette Co., W.Va.	1	34.76	61.39	3.85	1.92	.008	10,334	593	72.8	23.2	31.0	9.4
EAGLE	Fayette Co., W.Va.	Aver. 2	32.55	64.03	3.42	0.70	.004	10,627	612	74.1	23.4	27.7	8.0
		Max.	32.70	65.12	4.35	0.71	.004	10,850	652	74.3	23.6	27.8	8.6
		Min.	32.40	62.95	2.48	0.70	.003	10,403	572	73.9	23.2	27.6	7.4
	Kanawha Co., W.Va.	Aver. 5	32.44	63.72	3.84	0.72	.006	10,601	568	74.4	23.1	26.7	7.6
		Max.	33.34	65.50	5.08	0.78	.010	10,819	578	76.8	23.6	29.0	10.0
		Min.	30.06	61.92	3.09	0.63	.003	10,368	562	72.8	22.6	24.5	7.0
	Logan County, W.Va.	Aver. 4	31.94	63.80	4.26	0.66	.010	10,048	555	73.8	23.8	26.5	7.4
		Max.	32.68	65.67	5.14	0.74	.032	10,560	562	74.5	25.0	29.7	8.2
		Min.	30.27	62.60	3.64	0.55	.003	9,660	544	73.1	23.0	22.8	5.8
	Raleigh Co., W.Va.	Aver. 3	30.24	65.82	3.94	0.63	.004	10,234	569	75.5	22.1	23.9	8.1
		Max.	31.36	67.46	6.06	0.66	.006	10,562	576	76.1	23.4	25.0	9.2
		Min.	29.40	64.54	2.59	0.58	.003	9,668	564	75.0	21.2	22.8	7.4
NO. 2 GAS (POND CREEK)	Fayette Co., W.Va.	1	30.18	66.44	3.38	0.92	.032	10,318	583	76.1	22.6	22.4	7.6
	Logan Co., W. Va.	1	35.00	61.06	3.94	0.85	.002	10,082	564	71.7	25.4	28.5	8.4
	Pike County, Ky.	Aver. 2	31.82	61.00	7.18	0.80	.005	9,791	580	72.9	25.4	26.3	7.8
		Max.	32.53	61.18	7.70	0.89	.005	10,060	600	75.0	26.2	28.3	10.2
		Min.	31.12	60.81	6.66	0.70	.004	9,522	561	70.8	24.6	24.4	5.4
LOWER ELKHORN	Pike County, Ky.	Aver. 4	33.72	63.69	2.59	0.77	-	11,140	602	67.6	24.3	-	-
		Max.	33.86	64.81	3.85	1.04	-	11,973	612	68.5	25.0	-	-
		Min.	33.61	62.46	1.33	0.55	-	10,358	585	67.2	23.5	-	-
HARLAN	Harlan Co., Ky	Aver. 29	37.50	58.02	4.21	1.03	.005	10,752	-	70.7	31.8	29.3	7.8
		Max.	38.90	62.04	9.71	1.78	.022	11,390	-	71.6	36.4	35.8	9.6
		Min.	34.08	48.46	2.60	0.63	.001	9,570	-	69.6	28.6	22.0	4.4
CHILTON	Logan Co., W.Va.	Aver. 4	35.58	59.50	4.92	0.68	.004	10,639	572	71.2	25.4	26.4	9.0
		Max.	37.00	61.06	5.61	0.85	.006	10,900	591	72.9	29.2	28.5	10.6
		Min.	34.33	58.22	3.94	0.56	.002	9,900	562	70.0	22.6	24.6	6.8

NOTE: The above analyses are "tube tests" run at temperatures between 885°C and 900°C. The coke and ammonium sulphate yields reported are a little higher than would be obtained in actual oven operation, and the tar yield is about 40% lower than would be obtained in oven practice.

Data from Bureau of Mines and Private Records.

contents, without beneficiation, which can only rarely be equaled by coal from any other seam, and then with cleaning.

The ash fusing temperatures of the West Virginia coals are generally higher than those of the Virginia and Kentucky coals; the Chilton and Eagle seams, in this respect, are the best of any metallurgical coal.

TABLE 4.—*Reserves of High-volatile Metallurgical Coals, Southern West Virginia, as of January,* 1931

OWNED BY	SEAM	COUNTY	STATUS OF PROPERTY	ORIGINAL COAL NET TONS	MINED OUT NET TONS	REMAINING COAL		
						ACRES	AVER. THICK.	NET TONS
COMMERCIAL COMPANIES	POWELLTON	FAYETTE	Undeveloped	4,000,000	-	690	52"	4,000,000
		KANAWHA	Undeveloped	2,000,000	-	400	45"	2,000,000
		TOTAL	Undeveloped	6,000,000	-	1,090	49"	6,000,000
	EAGLE	BOONE	Undeveloped	66,200,000	-	10,500	56"	66,200,000
		FAYETTE	Developed	2,500,000	700,000	360	44"	1,800,000
			Undeveloped	14,300,000	-	2,520	50"	14,300,000
			Total	16,800,000	700,000	2,880	49"	16,100,000
		LOGAN	Developed	65,800,000	17,300,000	6,950	62"	48,500,000
		RALEIGH	Developed	29,300,000	4,000,000	4,160	54"	25,300,000
			Undeveloped	23,100,000	-	3,580	58"	23,100,000
			Total	52,400,000	4,000,000	7,740	56"	48,400,000
		TOTAL	Developed	97,600,000	22,000,000	11,470	58"	75,600,000
			Undeveloped	103,600,000	-	16,600	55"	103,600,000
			Total	201,200,000	22,000,000	28,070	57"	179,200,000
	NO. 2 GAS	FAYETTE	Developed	33,700,000	19,500,000	1,910	66"	14,200,000
			Undeveloped	11,700,000	-	1,900	55"	11,700,000
			Total	45,400,000	19,500,000	3,810	60"	25,900,000
		LOGAN	Undeveloped	58,000,000	-	9,910	52"	58,000,000
		MINGO	Developed	15,700,000	2,700,000	1,830	63"	13,000,000
		TOTAL	Developed	49,400,000	22,200,000	3,740	65"	27,200,000
			Undeveloped	69,700,000	-	11,810	53"	69,700,000
			Total	119,100,000	22,200,000	15,550	56"	96,900,000
	CHILTON	LOGAN	Developed	111,000,000	34,300,000	16,060	53"	76,700,000
			Undeveloped	17,600,000	-	3,770	52"	17,600,000
			Total	128,600,000	34,300,000	19,830	53"	94,300,000
	TOTAL ALL SEAMS	ALL COUNTIES	Developed	258,000,000	78,500,000	31,270	52"	179,500,000
			Undeveloped	196,900,000	-	33,270	54"	196,900,000
			Total	454,900,000	78,500,000	64,540	53"	376,400,000
CAPTIVE COMPANIES	POWELLTON	FAYETTE	Total	45,400,000	21,800,000	3,440	61"	23,600,000
		RALEIGH	"	7,400,000	-	1,340	49"	7,400,000
		TOTAL	"	52,800,000	21,800,000	4,780	58"	31,000,000
	EAGLE	FAYETTE	Total	27,800,000	6,300,000	3,300	58"	21,500,000
		LOGAN	"	20,700,000	4,800,000	2,440	58"	15,900,000
		RALEIGH	"	14,500,000	5,600,000	1,350	59"	8,900,000
		TOTAL	"	63,000,000	16,700,000	7,090	58"	46,300,000
	NO. 2 GAS	FAYETTE	Total	11,900,000	4,200,000	1,550	44"	7,700,000
		LOGAN	"	22,200,000	-	2,990	66"	22,200,000
		TOTAL	"	34,100,000	4,200,000	4,540	58"	29,900,000
	TOTAL ALL SEAMS	ALL COUNTIES	Total	149,900,000	42,700,000	16,410	58"	107,200,000
COMMERCIAL AND CAPTIVE COMPANIES	TOTAL ALL SEAMS	ALL COUNTIES	Total	604,800,000	121,200,000	80,950	54"	483,600,000

NOTES: Above Reserves do not include coal under 42" thick.
Chilton Seam - Reserves based on 60% Recovery.
All Other Seams - Reserves based on 75% Recovery.

BYPRODUCT ANALYSES

Not many byproduct analyses of coals have been published, although the U. S. Bureau of Mines is now making a study of all American gas coals for the American Gas Assn., and has published the data obtained for the Elkhorn coal and completed the work on the Chilton seam. These analyses are combined in Table 3 with data collected by the writer from many of the seams. Many of these samples were from outcrop openings which may not reflect the quality of the coal as well as mine samples. The difference is likely to be small, however, and aside from this fact the results given should be representative. The method used in these analyses gives tar yields that are much lower than those obtained in practical use in byproduct coke ovens; the other yields are comparable with those in practice.

TABLE 5.—*Reserves of High-volatile Metallurgical Coals, Kentucky, as of January,* 1931

OWNED BY	SEAM	COUNTY	STATUS OF PROPERTY	[illegible] COAL [illegible]	MINED OUT NET TONS	REMAINING COAL		
						ACRES	AVER. THICK.	NET TONS
COMMERCIAL COMPANIES	RODA (a)	HARLAN	Developed	33,600,000	4,400,000	5,300	49"	29,200,000
			Undeveloped	98,300,000	-	19,420	45"	98,300,000
			Total	131,900,000	4,400,000	24,720	46"	127,500,000
		LETCHER	Undeveloped	14,100,000	-	2,320	54"	14,100,000
		TOTAL	Developed	33,600,000	4,400,000	5,300	49"	29,200,000
			Undeveloped	112,400,000	-	21,740	46"	112,400,000
			Total	146,000,000	4,400,000	27,040	47"	141,600,[illegible]
	ELKHORN (b)	FLOYD	Developed	24,200,[illegible]	2,200,000	3,760	52"	22,000,000
			Undeveloped	47,200,[illegible]	-	8,740	48"	47,200,000
			Total	71,400,000	2,200,000	12,500	49"	69,200,000
		KNOTT	Developed	17,900,000	(e)	2,870	56"	17,900,000
			Undeveloped	16,400,000	-	3,100	47"	[illegible],400,000
			Total	34,300,000	-	5,970	51"	34,300,000
		LETCHER	Developed	289,800,000	56,800,[illegible]	33,380	62"	233,000,000
		PIKE	Developed	80,600,000	2,500,000	10,750	65"	78,[illegible],000
			Undeveloped	172,300,000	-	27,710	55"	172,[illegible],000
			Total	252,900,000	2,500,000	38,460	58"	250,400,000
		TOTAL	Developed	412,500,000	61,500,000	50,760	61"	351,000,000
			Undeveloped	235,900,000	-	39,550	53"	235,900,000
			Total	648,400,[illegible]	61,500,000	90,310	58"	586,900,000
	NO. 2 GAS (c)	MARTIN	Developed	10,100,000	1,500,000	1,350	57"	8,600,000
			Undeveloped	3,400,000	-	630	48"	3,400,000
			Total	13,500,000	1,500,000	1,980	54"	12,000,000
		[illegible]	Developed	41,200,000	16,100,000	4,130	54"	25,100,000
			Undeveloped	268,100,000	-	41,390	58"	268,100,000
			Total	309,300,000	16,100,000	45,520	57"	293,200,000
		TOTAL	Developed	51,300,000	17,600,000	5,[illegible]	55"	33,700,[illegible]
			Undeveloped	271,500,[illegible]	-	42,[illegible]	57"	[illegible]1,500,[illegible]
			Total	322,800,000	17,600,000	47,[illegible]	57"	305,200,[illegible]
	LOWER ELKHORN (d)	[illegible]	Developed	3,600,000	400,000	660	43"	3,200,000
			Undeveloped	103,600,000	-	17,600	52"	103,600,000
			Total	107,200,000	[illegible],000	18,260	52"	106,800,000
	HARLAN	HARLAN	Developed	160,400,000	44,700,000	20,540	50"	115,700,000
			Undeveloped	23,300,000	-	4,500	46"	23,[illegible],[illegible]
			Total	183,700,000	44,700,000	25,040	49"	139,000,000
	TOTAL ALL SEAMS	ALL COUNTIES	Developed	661,400,000	128,600,000	82,740	57"	532,800,000
			Undeveloped	746,700,000	-	125,410	53"	746,700,000
			Total	1,408,100,000	128,600,000	208,150	55"	1,279,500,000
CAPTIVE COMPANIES	RODA (a)	LETCHER	Total	25,500,000	-	4,200	54"	25,500,000
		HARLAN	"	122,900,000	38,000,000	14,800	51"	84,900,000
		TOTAL	"	148,400,000	38,000,000	19,000	52"	110,400,000
	ELKHORN (b)	FLOYD	Total	37,800,000	3,400,000	6,630	46"	34,400,000
		PIKE	"	80,300,000	8,700,000	13,530	47"	71,600,000
		KNOTT	"	28,100,000	1,900,000	4,570	51"	26,200,000
		TOTAL	"	146,200,000	14,000,000	24,730	48"	132,200,000
	NO. 2 GAS (c)	PIKE	Total	150,800,000	23,000,000	22,700	50"	127,800,000
	LOWER ELKHORN (d)	PIKE	Total	19,800,000	2,300,000	3,000	52"	17,500,000
	HARLAN	HARLAN	Total	116,800,000	7,100,000	17,400	56"	109,700,000
	TOTAL ALL SEAMS	ALL COUNTIES	Total	582,000,000	84,400,000	86,830	51"	497,600,000
COMMERCIAL AND CAPTIVE COMPANIES	TOTAL ALL SEAMS	ALL COUNTIES	Total	1,990,100,000	213,000,000	294,980	54"	1,777,100,000

NOTES: Above Reserves based on 75% Recovery and do not include coal under 42" thick.

(a) Known also as Darby, No. 5, Keokee, or Taggart Seam. Also the Upper Elkhorn Seam.

(b) Known also as Thick Elkhorn and in Pike Co. as Thick Elkhorn, Upper Elkhorn or Alma.

(c) Known as Pond Creek or Warfield in Pike Co., Ky. and Martin Co., Ky.

(d) Also known as No. 2 Gas and Imboden.

(e) Mining in adjacent county.

AREAS AND TONNAGES OF VARIOUS SEAMS

The figures in Tables 4, 5 and 6 were obtained by plotting on the county geological maps where these were available, or from private

TABLE 6.—*Reserves of High-volatile Metallurgical Coals, Virginia, as of January,* 1931

OWNED BY	SEAM	COUNTY	STATUS OF PROPERTY	ORIGINAL COAL NET TONS	MINED OUT NET TONS	REMAINING COAL		
						ACRES	AVER. THICK.	NET TONS
COMMERCIAL COMPANIES	RODA (TAGGART)	LEE	Developed	26,900,000	13,400,000	2,460	49"	13,500,000
		WISE	Developed	57,900,000	27,300,000	5,560	49"	30,600,000
			Undeveloped	28,500,000	-	5,510	46"	28,500,000
			Total	86,400,000	27,300,000	11,070	48"	59,100,000
		TOTAL	Developed	84,800,000	40,700,000	8,020	49"	44,100,000
			Undeveloped	28,500,000	-	5,510	46"	28,500,000
			Total	113,300,000	40,700,000	13,530	48"	72,600,000
	IMBODEN	LEE	Undeveloped	49,900,000	-	8,540	52"	49,900,000
		WISE	Developed	186,200,000	52,300,000	19,520	61"	133,900,000
		TOTAL	Total	236,100,000	52,300,000	28,060	58"	183,800,000
	TOTAL ALL SEAMS	ALL COUNTIES	Developed	271,000,000	93,000,000	27,540	57"	178,000,000
			Undeveloped	78,400,000	-	14,050	50"	78,400,000
			Total	349,400,000	93,000,000	41,590	55"	256,400,000

NOTE: Above Reserves based on 75% Recovery and do not include coal under 42" thick.

sources, all of the thicknesses measured at the outcrop openings and in the mines and found by drilling, in each of the seams, and then drawing the outline of the areas in which each seam is 42 in. thick, or over, of workable coal of the ash and sulfur contents specified. These areas were then measured by planimeter and the average thickness was calculated. Assuming a yield of 1350 net tons per acre foot, about 75 per cent recovery, these quantities gave the tonnages of recoverable coal in each seam originally of coal of the desired specifications.

TABLE 7.—*Summary of Reserves of High-volatile Metallurgical Coals, Southern West Virginia, Kentucky and Virginia, as of January, 1931*

SEAM	COMMERCIAL OR CAPTIVE	STATE	ORIGINAL COAL NET TONS	MINED OUT NET TONS	REMAINING COAL		
					ACRES	AVER. TH.	NET TONS
RODA (a)	Commercial	Kentucky	146,000,000	4,400,000	27,040	46"	141,600,000
	"	Virginia	113,300,000	40,700,000	13,530	48"	72,600,000
	Captive	Kentucky	148,400,000	38,000,000	19,000	52"	110,400,000
	Total	Total	407,700,000	83,100,000	59,570	49"	324,600,000
ELKHORN (b)	Commercial	Kentucky	648,400,000	61,500,000	90,310	58"	586,900,000
	Captive	"	146,200,000	14,000,000	24,730	48"	132,200,000
	Total	"	794,600,000	75,500,000	115,040	55"	719,100,000
POWELLTON	Commercial	West Va.	6,000,000	-	1,090	49"	6,000,000
	Captive	"	52,800,000	21,800,000	4,780	58"	31,000,000
	Total	"	58,800,000	21,800,000	5,870	56"	37,000,000
EAGLE	Commercial	West Va.	201,200,000	22,000,000	28,070	57"	179,200,000
	Captive	"	63,000,000	16,700,000	7,090	58"	46,300,000
	Total	"	264,200,000	38,700,000	35,160	[illegible]7"	225,500,000
NO. 2 GAS (c)	Commercial	West Va.	119,100,000	22,200,000	15,550	55"	96,900,000
	"	Kentucky	322,800,000	17,600,000	47,500,	57"	305,200,000
	Captive	West Va.	34,100,000	4,200,000	4,540	58"	29,900,000
	"	Kentucky	150,800,000	23,000,000	22,700	50"	127,800,000
	Total	Total	626,800,000	67,000,000	90,290	55"	559,800,000
LOWER ELKHORN (d)	Commercial	Kentucky	107,200,000	400,000	18,260	52"	106,800,000
	Captive	"	19,800,000	2,300,000	3,000	52"	17,500,000
	Total	"	127,000,000	2,700,000	21,260	52"	124,300,000
HARLAN	Commercial	Kentucky	183,700,000	44,700,000	25,040	49"	139,000,000
	Captive	"	116,800,000	7,100,000	17,400	56"	109,700,000
	Total	"	300,500,000	51,800,000	42,440	52"	248,700,000
CHILTON	Commercial	West Va.	128,600,000	34,300,000	19,830	53"	94,300,000
IMBODEN	Commercial	Virginia	236,100,000	52,300,000	28,060	58"	183,800,000
TOTAL ALL SEAMS	Commercial	All States	2,212,400,000	300,100,000	314,280	54"	1,912,300,000
	Captive	"	731,900,000	127,100,000	103,240	52"	604,800,000
	Total	"	2,944,300,000	427,200,000	417,520,	54"	2,517,100,000

NOTES: Above Reserves do not include coal under 42" Thick.

Chilton Seam - Reserves based on 60% Recovery.

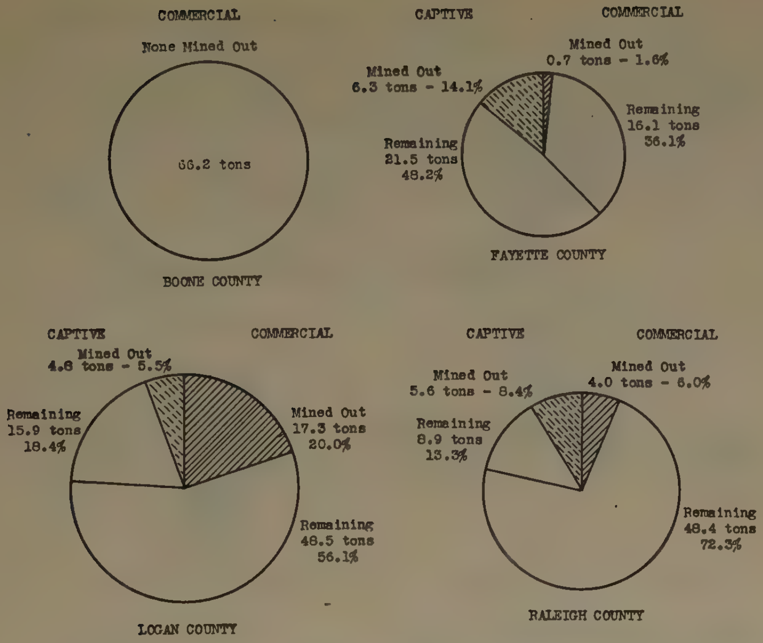
All Other Seams - Reserves based on 75% Recovery.

(a) Known also as Darby, No. 5, Keokee or Taggart. (Also the Upper Elkhorn Seam).

(b) Known also as Thick-Elkhorn and Upper Elkhorn or Alma.

(c) Known also as Pond Creek or Warfield.

(d) Known also as No. 2 Gas and Imboden.



Tonnage figures indicate millions of net tons of recoverable coal, over 42" thick, based on 75% recovery.

FIG. 8.—COAL RESERVES, EAGLE SEAM, WEST VIRGINIA.

Tonnage figures indicate millions of net tons recoverable coal, over 42" thick, based on 75% recovery for Powellton Seam and 60% recovery for Chilton Seam.

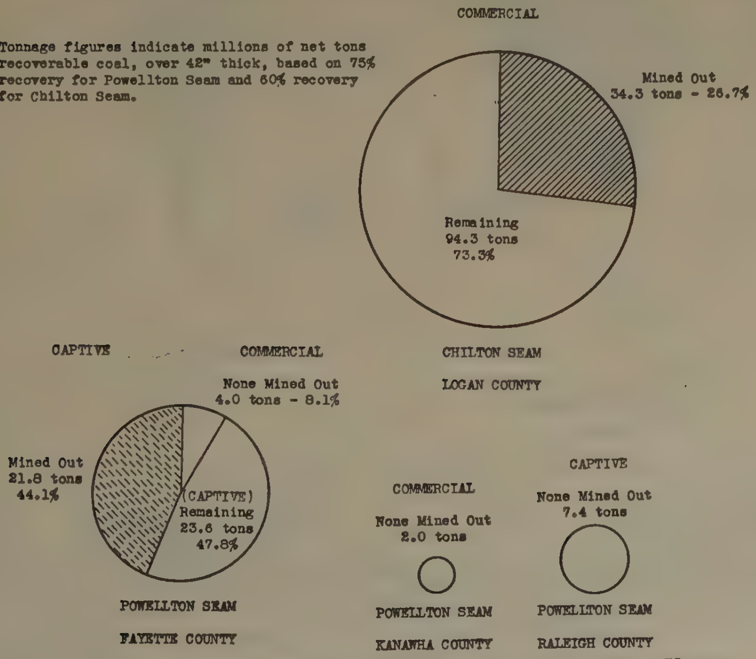
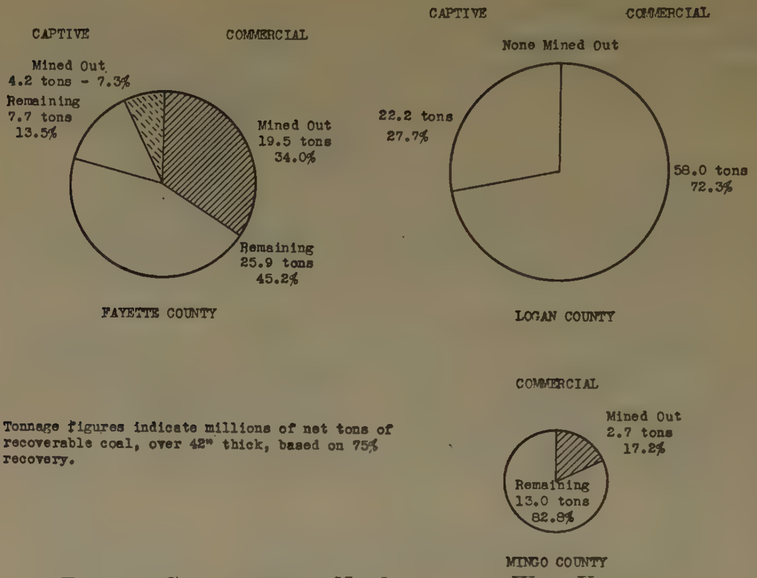
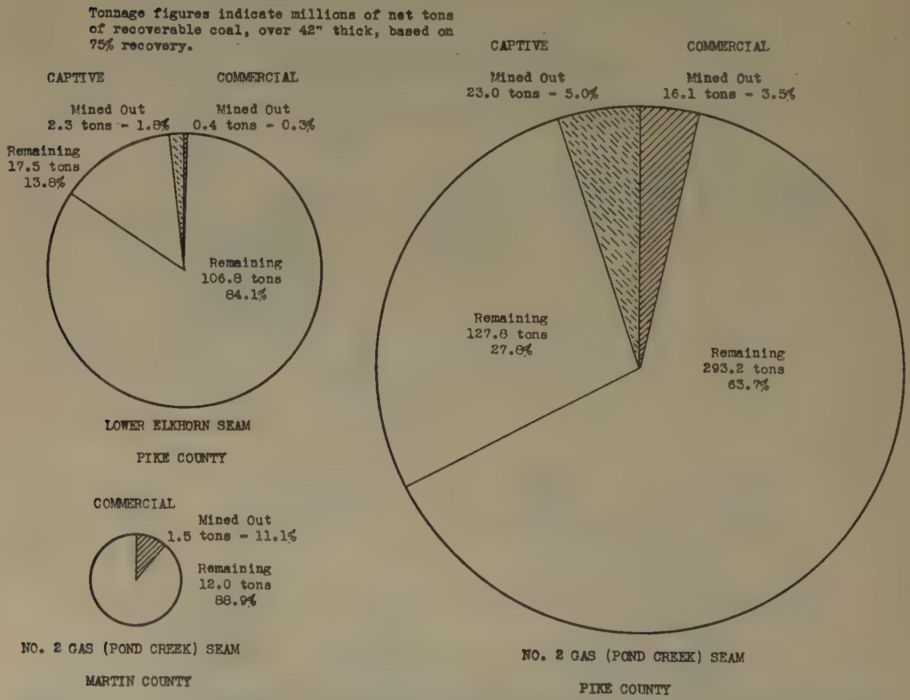


FIG. 9.—COAL RESERVES, POWELLTON AND CHILTON SEAMS, WEST VIRGINIA.



Tonnage figures indicate millions of net tons of recoverable coal, over 42" thick, based on 75% recovery.

FIG. 10.—COAL RESERVES, NO. 2 GAS SEAM, WEST VIRGINIA.



Tonnage figures indicate millions of net tons of recoverable coal, over 42" thick, based on 75% recovery.

FIG. 11.—COAL RESERVES, NO. 2 GAS (POND CREEK) AND LOWER ELKHORN SEAMS, KENTUCKY.

QUANTITIES MINED OUT

The tonnages mined, as of Jan. 1, 1931, were taken from the records of the mining departments of West Virginia and Kentucky and from records of the operators' associations of Virginia, these being the best data available. In some cases, where one plant mines several seams, it was necessary to prorate the shipments between seams in accordance with the information at hand; and in a few mines coal mined in one county is shipped from a tipple in another one. The figures for seams and counties are not therefore exact, but the discrepancies are small. In West Virginia and Kentucky we were able to classify the coal as working, stopped working and undeveloped. The first item is for developed mines now operating, the second is developed properties which have been shut

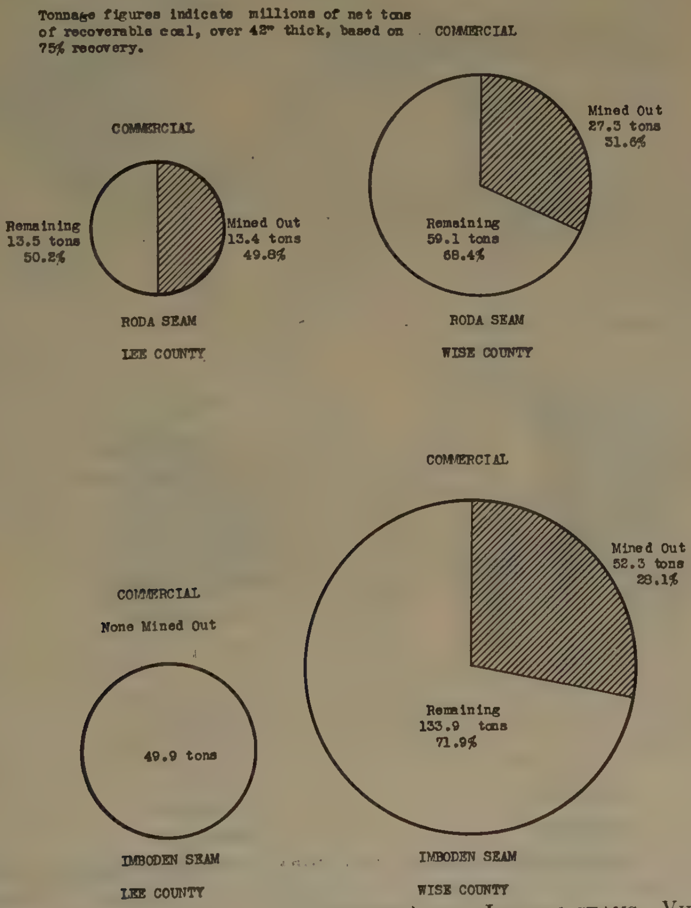


FIG. 12.—COAL RESERVES, RODA (TAGGART) AND IMODEN SEAMS, VIRGINIA.

down on account of economic reasons, while the third is for properties not yet opened. In Virginia this classification was not possible, and we could only separate the developed and undeveloped properties. In West Virginia and Kentucky we were also able to classify the coal between commercial coal being shipped on the open market and captive coal being shipped to the owners. In Virginia, so far as we know, there are only commercial mines.

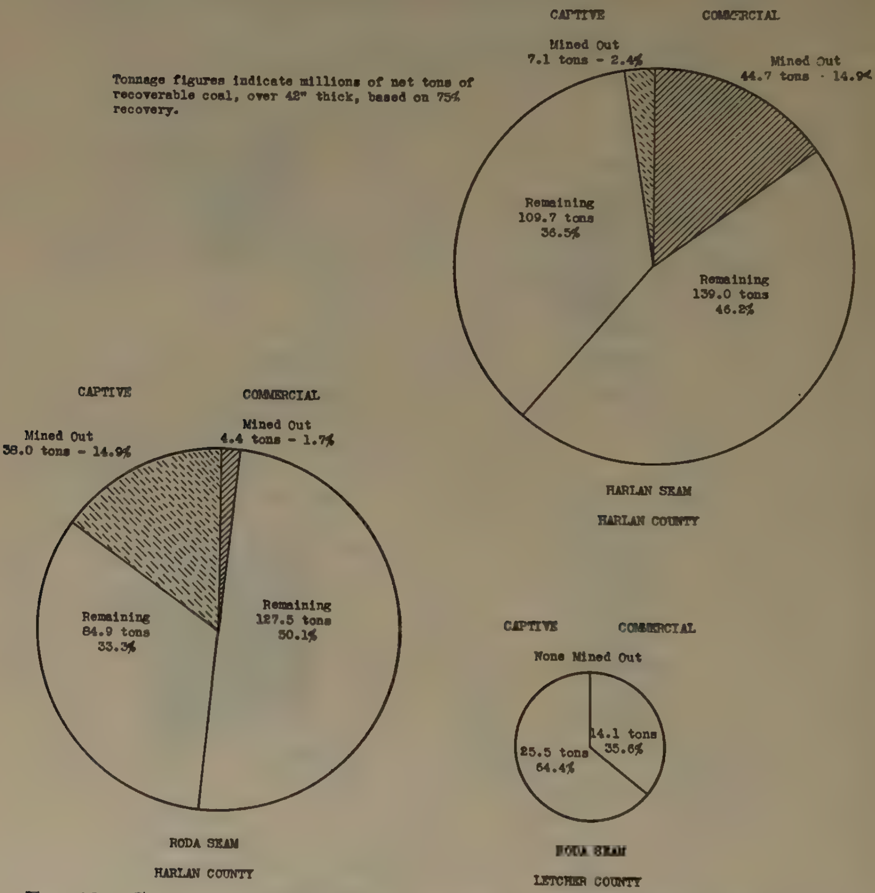
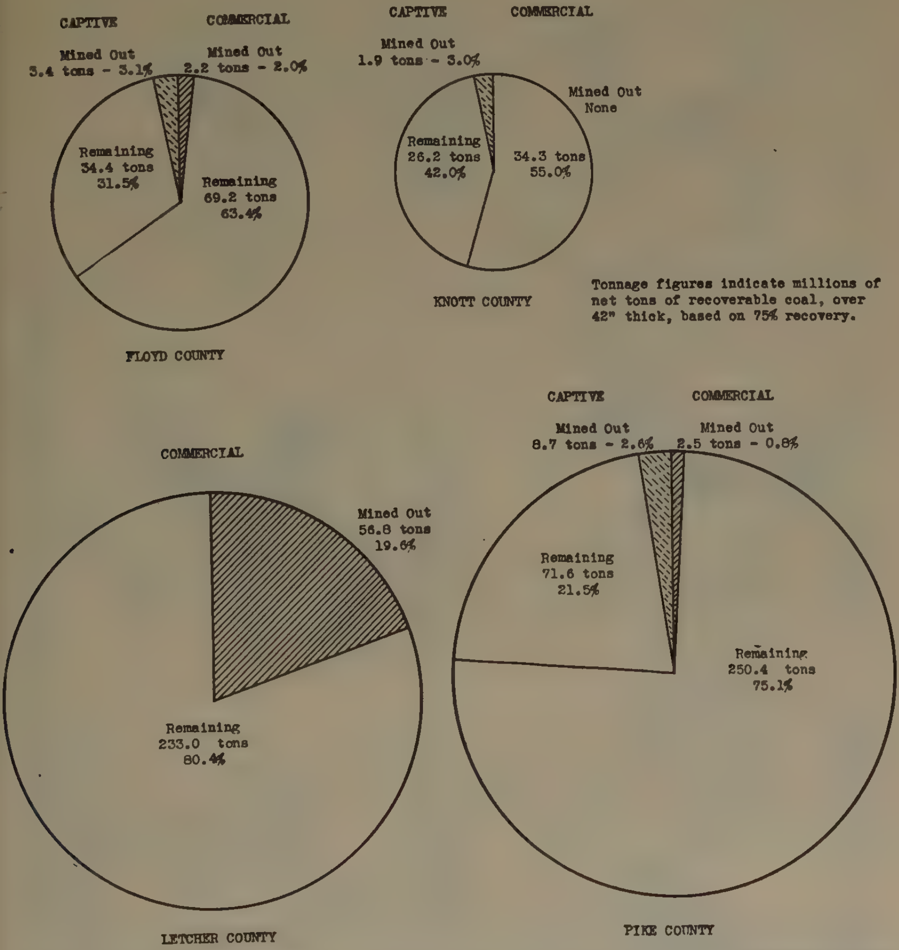


Fig. 13.—Coal reserves, Harlan and Roda (Taggart) seams, Kentucky.

Quantities of Coal Remaining

By deducting the amount mined out from the original recoverable coal we obtain the amounts remaining, and these for counties and seams in acres, tons and average thicknesses are all shown in Tables 4, 5 and 6 for the various states. A summary for the three states is shown in Table 7. A graphical representation of these figures is shown in Figs. 8 to 16 inclusive.



Tonnage figures indicate millions of net tons of recoverable coal, over 42" thick, based on 75% recovery.

FIG. 14.—COAL RESERVES, UPPER ELKHORN OR ALMA AND THICK ELKHORN SEAMS, KENTUCKY.

In considering these data it must be remembered that in each seam the development has been in the thickest and best coal, and that the reserves of undeveloped coal are largely of the thinner and poorer qualities of the seam from a mining standpoint, although the remaining coal, under the specifications mentioned, is still better than that usually available elsewhere.

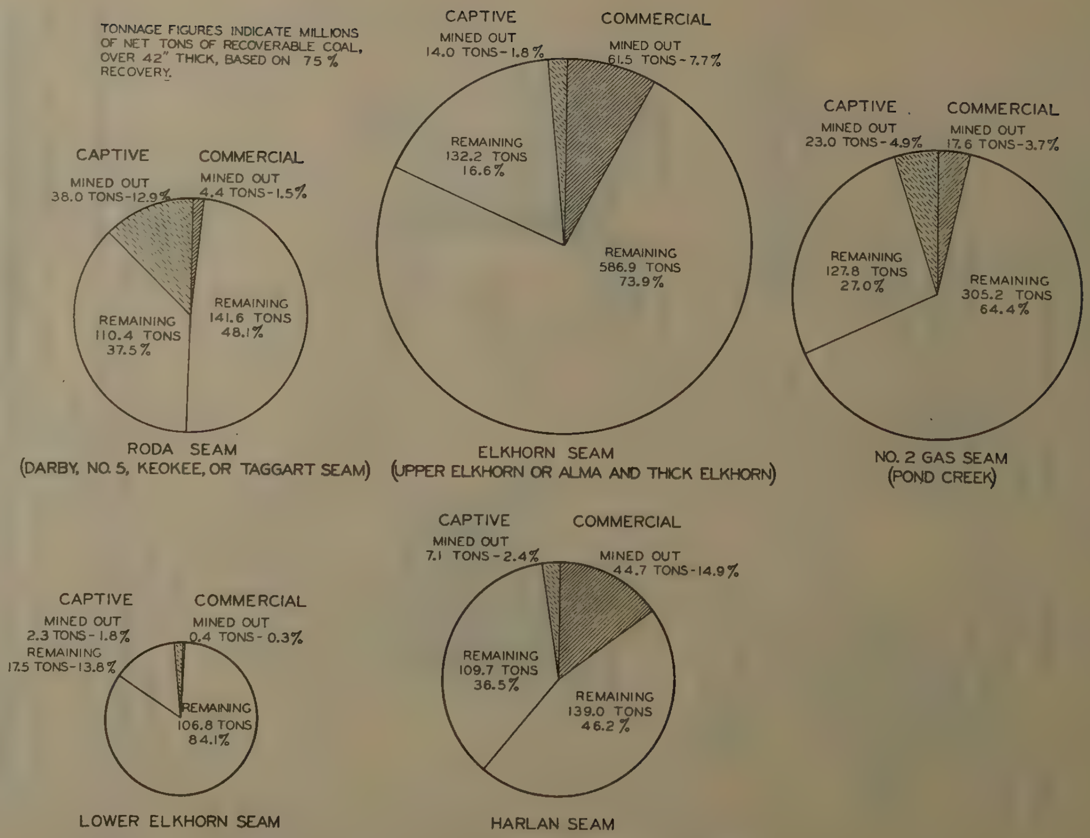
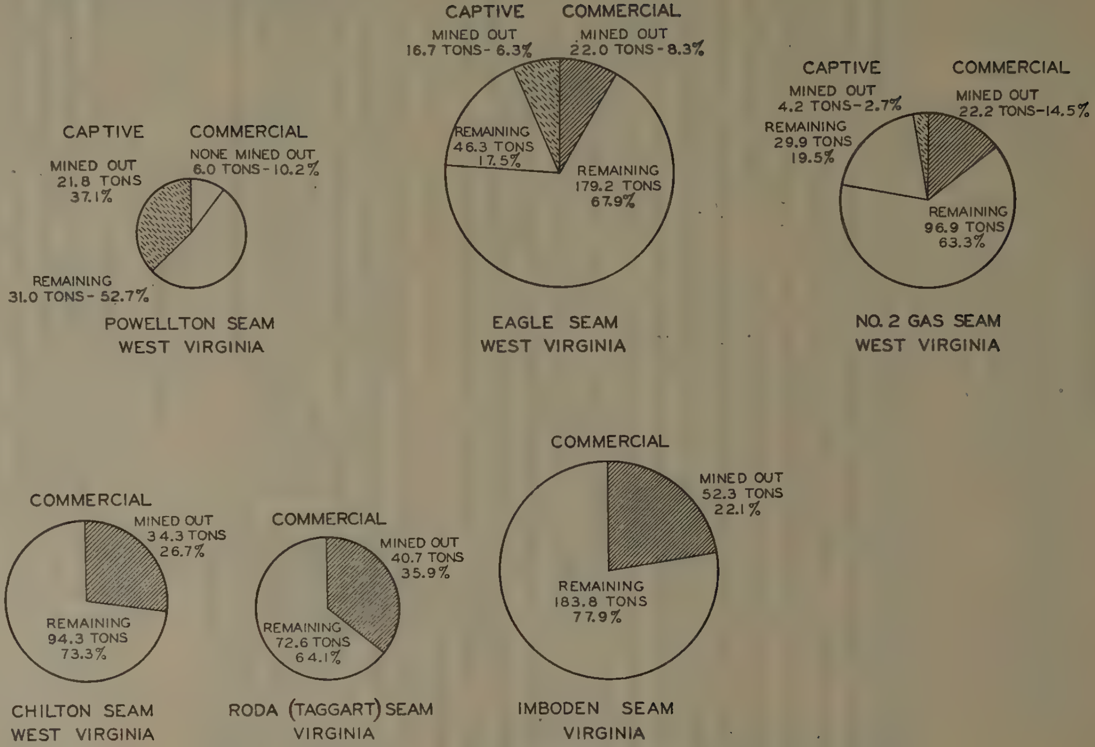


FIG. 15.—SUMMARY OF COAL RESERVES, KENTUCKY.



TONNAGE FIGURES INDICATE MILLIONS
OF NET TONS OF RECOVERABLE COAL,
OVER 42" THICK, BASED ON 60%
RECOVERY FOR CHILTON SEAM AND
75% RECOVERY FOR ALL OTHER SEAMS.

Fig. 16.—Summary of coal reserves, West Virginia and Virginia.

Preliminary Report on Unit Coal-Specific Gravity Curves of Illinois Coals

By L. C. McCabe,* D. R. Mitchell,† and G. H. Cady,‡ Urbana, Ill.

(New York Meeting, February, 1934)

Abstract§

While running certain float-and-sink tests, unit coal calorific values were determined for gravity fractions of a series of Illinois coals from several districts in the southern part of the state. When these values were plotted against specific gravity what appear to be characteristic curves of somewhat different shape were obtained for each district represented. This in itself seems noteworthy. The shape of the curves seems to indicate that the lightest fraction of the coal, which inspection shows to be largely vitrain, suffers the least deterioration in storage.

It was also observed that on dilution curves established with dry ash and B.t.u. values the fractions of 1.35, 1.40 and 1.50 sp. gr., which constitute the greater weight per cent of a given coal sample, fall more frequently on a straight dilution line than do the lighter or heavier fractions of the same sample. In most instances there is a marked steepening between the 1.35 and the fraction of lighter gravity. In general the curve increases in steepness between the fractions of 1.50 and 1.70 specific gravity.

The variations in the unit coal-specific gravity curves as well as those observed in dilution curves which appear to be pertinent to this problem are: (1) variation in coal substance with change in specific gravity, (2) variation in kind and quantity of mineral matter with the specific gravity of the fraction, (3) factors of analytical procedure which tend to introduce error in analysis, particularly in high-ash and high-sulfur fractions.

DISCUSSION

(*A. C. Callen presiding*)

E. Stansfield,|| Edmonton, Alta.—The necessity of avoiding oxidation in grinding and in drying should be emphasized. My assumptions as to pure coal coincide with the author's but most of my curves are nearly straight lines. The low sulfur content

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§ Published by permission of the Chief, Illinois State Geological Survey. The entire paper will be published as Illinois Geol. Survey *Report of Investigations No.* 34.

|| Chief Chemical Engineer, Research Council of Alberta.

of Alberta coals simplifies the problem here. There is a notable difference in method, however, as we employ several samples and separate each into a float and sink, while the authors float successive portions from the same sample. To get a correct pure coal value it is necessary to avoid too low an ash, as it is safer to extrapolate than to risk undue fractionation of vitrain, for example. We plot not only calorific values but also complete proximate and ultimate analyses against the ash. It is noteworthy that although the moisture is always higher in the pure coal by the Parr calculations, the curves sometimes show rising and sometimes falling moisture as the ash decreases.

Although we plot calorific value against ash on the dry basis for record and study, we provide the operators with a set of curves that give the calorific value and ash for several different moisture contents in the neighborhood of the typical moisture for the coal.

A. C. FIELDNER,* Washington, D. C.—A few coals have been examined at Pittsburgh by ultimate and proximate analyses of the different bands, and the results of the ultimate analysis agreed within the limits of error, while the volatile matter and fixed carbon varied 2 per cent. In other cases both the ultimate and proximate analyses differed.

W. A. SELVIG,† Pittsburgh, Pa.—The difficulty lies principally with the volatile matter, which may vary considerably in different bands of the coal bed.

L. C. MCCABE (written discussion).—It is possible that the banded ingredients of the coals concerned in this report exhibit differences that are not marked in the coals of higher rank. Certain important differences were brought out in some 60 band samples, which were carefully collected and analyzed during the past two years. The analyses of fusain show great variability in moisture and ash content. Within the same mine moisture values vary from 8.99 to 22.8 per cent and the ash from 4.86 to 16.91 per cent. This variability in moisture and ash content is due to the porous nature of the fusain; air-drying loss is, for the same reason, very high. Other noteworthy chemical peculiarities of fusain are a very high content of fixed carbon, which in the dry, ash-free coal is about 75 per cent, so that the fuel ratio $\frac{(FC)}{(VM)}$ is commonly more than 3.0. The unit coal value of fusain is fairly constant at about 15,000 B.t.u., although there are erratic and unexpected variations from this value.

Vitrain is generally characterized by less ash than the other ingredients (0.49 per cent ash samples have been collected) and a relatively low unit coal value. The composition of clarain does not differ greatly from the raw coal. The fuel ratio of clarain and vitrain is much lower than that of fusain, but in samples from the same mine the fuel ratio of clarain is found to be lower than that for vitrain. In general there is an increase in the fuel ratio for the bands as well as for the raw coal southeastward across the southern part of the state. The ultimate analyses show that the clarain contains more available hydrogen than vitrain.

A petrographic analysis of the float-and-sink fractions from one of the mines under consideration showed a recovery of 66.8 per cent vitrain in the fraction of 1.30 sp. gr., 90.7 per cent of the fraction being vitrain. The fractions of 1.35 to 1.70 sp. gr. contained 95.4 per cent of the clarain. It is believed that in the Illinois coals the deviation of the light fraction from a straight line in the dilution curve indicates real differences between the vitrain and the clarain, which are not brought out by extrapolation when the low-ash vitrain is not isolated.

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Unit Coal as a Basis of Coal Standardization as Applied to Illinois Coals

By Gilbert H. Cady* and O. W. Rees,† Urbana, Ill.

(New York Meeting, February, 1934)

Abstract‡

Unit coal calorific values represent a close approach to pure coal values; they possess characteristics theoretically inherent in the latter values. They are essentially uniform in the same seam for areas possessing essentially uniform geological environment. They are higher in older than in younger coals in most cases, and in deeper than in shallower coals. In general, they display characteristics closely in accordance with expectation assuming that they are essentially indices of rank. Because it possesses these characteristics, the unit coal value is essentially a standard value, characteristic of each bed in each local region. Hence it can be used as a basis for determining the accuracy of results in newly established laboratories, or when new equipment is installed, or in checking the results of different laboratories, or even in determining the character of coal as mined or prepared if ash, moisture and sulfur values are known.

DISCUSSION

(*A. C. Callen presiding*)

R. P. Hudson,§ Wayland, Ky. (written discussion).—One of the interesting observations made by the authors is that there is a more or less definite correlation between unit coal calorific values and the geological environment of the coal bed. It is very interesting to note that, in general, the unit B.t.u. value increases with the increase in the probable original depth of burial, and that each seam tends to increase in calorific value toward the areas of diastrophic instability.

It is generally conceded that Parr's formula, for determining unit coal values, is about as accurate as any formula subsequently proposed for the same purpose. In his formula, Professor Parr did not differentiate between the sulfur present as iron pyrites and that present as organic sulfur. However, the practical application of the formula has shown that this is of little consequence.

I agree with the authors that unit coal calorific values are useful in determining the accuracy of calorimetric determinations, and in determining the character of the

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[†] Associate Chemist and Head of Analytical Division, Illinois State Geological Survey.

[‡] Presented with the approval of the Chief, Illinois State Geological Survey. The paper will be published in Illinois Geol. Survey *Report of Investigations No.* 32.

[§] Chemist, Elk Horn Coal Corporation.

coal as mined, if other analytical data are at hand. It is obvious that any formula that indicates the heating value of the coal will, in its reversed form, indicate the heating value of the coal in the condition in which it was received in the laboratory.

Our company operates several mines, covering a rather wide area in eastern Kentucky. All of these mines are operating in the Elkhorn coal bed, but some are in the No. 1 bed and others mine both the No. 1 bed and the No. 2 bed, of Elkhorn coal. Unit coal calorific values have been established for each mine. These values, although varying in each seam and from mine to mine, are quite constant for any one particular operation. Therefore we have discontinued calorimetric determinations on the mined coal. We have checked the calculated and determined values many times and the agreement was so close that it is obviously of no value to make B.t.u. determinations in ordinary routine work.

Unit coal values are useful for other purposes. At times, relatively small coal samples are sent to the laboratory, by customers or coal salesmen, for identification as to mine origin. After analyzing the coal, it is quite easy, by the aid of unit coal values, to determine from which mine the coal originated.

Kentucky coals—at least those of Eastern Kentucky—probably will not show such a definite correlation of unit coal calorific values, for each seam, within such a narrowly restricted geographical district as the authors found for Illinois coals. They seem to have proved that, for Illinois coals, county average B.t.u. values, for each seam, may be used without causing any practical error. With many other coals, however, narrowly restricted geographical averages probably would not work out so well. This is not, of course, a criticism of the authors' work, as it is confined entirely to Illinois coals.

The Technical Committee on Scientific Classification, of the American Standards Association Committee on Coal Classification, has proposed the grouping of high-volatile coals according to rank by reference to fixed carbon and B.t.u., on the basis of coal containing its natural bed moisture but free from mineral matter. This seems to be a reasonable proposal, because the normal bed moisture is obviously a part of the coal, rather than extraneous material. However, since the normal bed moisture is, in many Illinois coals, a highly variable factor, the calculation of unit coal calorific values on the moisture-free basis would seem to serve the purpose of the authors better than values obtained on the moist unit coal basis.

T. A. HENDRICKS,* Washington, D. C.—Dr. Cady's thesis that unit coal values will reveal stratigraphic differences should be modified to the extent of saying that, for this to be possible, the coals must have been of the same original material. In southeastern Oklahoma the McAlester bed is 1300 ft. above a lower bed of coal but has a higher B.t.u. value. This is attributed to the probability that the lower bed is of different original material. Microscopic study is needed to determine this. As to Mr. Hudson's point that county averages apply in Illinois, this would not be true in many other areas.

R. D. HALL,† New York, N. Y.—What are the ages of the anticlines?

G. H. CADY.—The LaSalle anticline probably began during the Pennsylvanian period but the DeQuoin anticline is younger. I agree with Mr. Hendricks that when the upper coals have higher calorific value than lower coals, they are probably of different types, otherwise there is a close relation between the stratigraphic position and the unit coal values.

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* U. S. Geological Survey.

† Engineering Editor, *Coal Age.*

Mineral Matter in Coal—A Preliminary Report

By A. W. Gauger,* E. P. Barrett* and F. J. Williams,* State College, Pa.

(New York Meeting, February, 1934)

Coal as mined contains varying quantities of inorganic components (mineral matter) which, on combustion, produce the residue known as ash. It has long been realized that the weight of this residue does not equal the weight of the mineral matter originally in the coal, because of the chemical changes that take place during the ashing process. Moreover, the ordinary ash analysis presents the ash as a mixture of oxides with little or no clue to the identity of the mineral species originally present in the coal. It is only for pyritic sulfur that an attempt is made to determine a mineral component.

So far as the authors are aware, there are practically no experimental data in the literature as to the percentages of the different mineral species present in coal. The oft-quoted table of Marson and Cobb[1] cannot rightly be considered as data, since the authors merely state that the table "shows the probable forms in which these ash constituents exist in coal." In view of the attempts to calculate analyses and determinations of the calorific equivalent of coal to the mineral-matter-free basis, as well as the importance of the mineral species in clinkering, combustion and coking, it seems strange that so few data are available on this phase of coal technology.

For example, numerous empirical corrections for use in the scientific classification of coal have been suggested to compensate for the losses from the mineral matter during ignition of the coal to ash. These include multiplication of the ash percentage by various factors such as $1\frac{1}{10}$ and $1\frac{1}{8}$, the use of the Parr unit coal formula ($1.08\ A + \frac{22}{40}\ S$, where S = total sulfur) as well as certain modifications of the Parr formula that attempt to correct for the organic sulfur present in the coal. Although many experiments have been made to determine the loss on ignition of the clays associated with the coal measures and correction

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Manuscript received at the office of the Institute Dec. 9, 1933.

* Mineral Industries Experiment Station, The Pennsylvania State College.

[1] C. B. Marson and J. W. Cobb: Influence of the Ash Constituents in the Carbonization and Gasification of Coal, with Especial Reference to Nitrogen and Sulfur, Part I—Preparation and Preliminary Examination of Special Cokes. *Gas Jnl.* (London), (1925) **171,** 39–46.

factors based on these experiments have been suggested,[2] there is some evidence, as pointed out by Mott and Wheeler,[3] that the shales differ considerably from the mineral matter separated by flotation from the coal itself.

Recently the Mineral Industries Experiment Station of The Pennsylvania State College had occasion to examine seven samples of coal and three samples of washery refuse for the purpose of obtaining an approximate knowledge of the minerals present. This examination consisted of concentrating the inorganic material by means of float and sink separations followed by petrographic studies of the separated materials. Successive separations were made at specific gravities of 1.6, 2.0, and 2.6, so that each sample was divided into four fractions. The plus 1.6 fractions were not studied because the large amount of carbonaceous matter completely obscured the minerals which were present. Even the plus 2.6 fractions contained enough carbonaceous material to make the petrographic examination rather difficult.

TABLE 1.—*Results of Petrographic Study of Ten Samples*

Name	Formula	Remarks
Pyrite	FeS_2	Found in all samples
Kaolin minerals	$Al_2O_3.2SiO_2.xH_2O$	Found in all samples
Chlorites:		
Prochlorite[a]	$2FeO.2MgO\ Al_2O_3.2SiO_2.2H_2O$	Found in all samples
Penninite[a]	$5(MgFe)O.Al_2O_3.3SiO_2.2H_2O$	Found in 3 samples
Muscovite	$KNaO.3Al_2O_3.6SiO_2.2H_2O$	Found in all samples
Calcite	$CaCO_3$	Found in 9 samples
Quartz	SiO_2	Found in all samples
Diaspore	$Al_2O_3.H_2O$	Found in 2 samples
Limonite	$2Fe_2O_3.3H_2O$	Found in 7 samples
Magnetite	Fe_3O_4	Found in 7 samples
Gypsum	$CaSO_4.2H_2O$	Found in all samples
Rutile	TiO_2	Found in 2 samples
Hematite	Fe_2O_3	Found in 1 sample
Tourmaline	Not constant; a complex aluminum borosilicate	Found in 2 samples
Siderite	$FeCO_3$	Found in 2 samples
Zircon	$ZrSiO_4$	Found in 2 samples
Garnet	$Ca_3Al_2Si_3O_{12}$	Found in 1 sample

[a] These two forms not positively identified but two different chlorites were present and they were tentatively identified as penninite and prochlorite.

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[2] F. V. Tideswell and R. V. Wheeler: Pure Coal as a Basis for Classification. *Trans.* A.I.M.E. (1928) **76**, 200–214.

A. C. Fieldner and W. A. Selvig: Present Status of Ash Corrections in Coal Analysis. *Trans.* A.I.M.E. (1930) **88**, Coal Div., 597–611.

[3] R. A. Mott and R. V. Wheeler: *Fuel in Sci. and Prac.* (1927) **6**, 416.

The minerals recognized in the samples are listed in Table 1. It should be emphasized that these are by no means all of the minerals present. They are only those positively identified.

Ash analyses for several of these samples were available. These analyses were rationalized with the aid of the information obtained petrographically and the results for three samples are presented in Table 2. The procedure used for rationalization will be illustrated later.

TABLE 2.—*Ash Analyses and Calculated Mineral Analyses*

	Ash Analysis, Per Cent in Coal				Mineral Analysis, Per Cent in Coal		
	Sample No.				Sample No.		
	1	2	3		1	2	3
SiO_2	3.92	2.82	2.58	Pyrite	5.40	3.01	0.38
Fe_2O_3	3.73	2.13	0.31	Prochlorite	0.29	0.12	0.12
Al_2O_3	2.68	2.44	2.20	Calcite	0.34	0.28	0.19
CaO	0.19	0.16	0.11	Muscovite	3.60	1.80	0.20
MgO	0.05	0.02	0.02	Kaolin	3.03	4.25	5.27
KNaO	0.36	0.18	0.02	Quartz	0.76		
				Diaspore		0.05	
				Limonite	0.06	0.09	0.13
Total	10.93	7.75	5.24		13.48	9.60	6.29
Parr noncoal					13.85	9.63	5.98
Ratio mineral matter to ash					1.23	1.24	1.20
Ratio (M.M. − pyrite) to (Ash − Fe_2O_3 equiv. of pyrite)					1.10	1.14	1.18

Assuming that in general the minerals present will be much the same in species throughout a given seam, although no doubt varying widely in relative quantities, a similar rationalization was applied to five sets of data including original, float and sink samples, taken from the work of Nicholls and Selvig, of the U. S. Bureau of Mines.[4] These data are given in Table 3. The procedure used to rationalize the ash analyses is outlined below:

1. The ash analysis on the 100 per cent basis is calculated to the dry-coal basis.
2. The percentage of each constituent is converted to mols by dividing by the respective molecular weights.
3. Pyritic sulfur is removed as mols pyrite.
4. Fe_2O_3 corresponding to pyrite is removed by subtracting one-half of the pyrite.

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[4] P. Nicholls and W. A. Selvig: Clinker Formation as Related to the Fusibility of Coal Ash. U.S. Bur. Mines *Bull.* 364 (1932) 5–14.

TABLE 3.—*Ash Analyses Calculated to Minerals*

Separations made at 1.35 specific gravity

Mineral	Sample A			Sample C			Sample E			Sample R			Sample RW		
	Original	Float	Sink	Original	Float	Sink	Original	Float	Sink	Original	Float	Sink	Original	Float	Sink
Pyrite	0.36	0.08	2.08	2.10	0.32	8.18	1.05	0.20	7.01	2.66	0.06	10.31	1.91	0.13	8.16
Gypsum	0.11	0.05	0.12	0.16	0.05	0.32	0.05	0.05		0.16	0.05	0.75	0.11	0.05	0.43
Apatite	0.65	0.03	2.27	0.03	0.01	0.08	0.02	0.01	0.04	0.07	0.04	0.18	0.07	0.04	0.16
Calcite		0.05		0.09	0.19		0.14	0.15	0.18	0.11	0.18		0.13	0.17	
Prochlorite	0.21	0.14	0.47	0.31	0.14	0.52	0.10	0.09	0.30	0.23	0.08	0.17	0.14	0.08	0.29
Limonite	0.03		0.43	0.22		0.84	0.10		0.94	0.50	0.05	2.10	0.26	0.07	1.14
Rutile	0.11	0.06	0.28	0.11	0.06	0.32	0.07	0.06	0.27	0.12	0.07	0.27	0.10	0.07	0.22
Muscovite	0.89	0.37	2.81	1.10	0.35	4.13	0.65	0.39	2.70	1.22	0.93	3.26	0.94	0.60	2.63
Kaolin	5.13	1.85	20.83	5.04	2.08	17.34	3.86	1.91	16.61	6.05	2.17	15.79	5.71	2.62	15.36
Quartz				0.76	0.13	3.24	0.24	0.06	1.78	0.18		1.26	0.18		1.06
Diaspore	0.36	0.31	0.60								0.09			0.07	
Total M.M.	7.85	2.94	29.89	9.92	3.33	34.97	6.28	2.92	29.83	11.30	3.72	34.09	9.55	3.90	29.45
Ash per cent	7.1	2.9	26.0	8.5	2.9	29.2	5.4	2.5	25.0	9.4	3.3	27.4	8.0	3.5	24.0

5. Sulfate sulfur is removed as gypsum and a corresponding amount of CaO removed.

6. P_2O_5 is calculated to apatite (mols apatite = ⅔ mols P_2O_5).

7. CaO corresponding to five times apatite is removed as apatite.

8. Any remaining CaO is calculated to calcite.

9. MgO is removed as prochlorite and corresponding amounts of Fe_2O_3, Al_2O_3 and SiO_2 removed.

10. Any remaining Fe_2O_3 is calculated to limonite (½ Fe_2O_3 equivalent to limonite).

11. TiO_2 is calculated as rutile.

12. Muscovite is calculated from K_2O and Na_2O and corresponding quantities of SiO_2 and Al_2O_3 are removed.

13. Balance of Al_2O_3 and SiO_2 are calculated to kaolinite and any excess of SiO_2 to quartz. In case of excess of Al_2O_3 it is considered as diaspore.

14. The molal composition is converted into weight composition by multiplying the respective number of mols by the corresponding molecular weights.

15. The sum of the weights of minerals present gives the total weight of mineral matter in 100 parts of the coal.

To interpret this outline correctly, it is essential to bear in mind that from step 3 to step 13, inclusive, the unit of quantity used is the mol, not the mol per cent. A detailed calculation for one sample, coal A, is shown in Table 4.

As stated before, many attempts have been made to calculate the calorific equivalent of "pure coal"; i. e., coal free from water and mineral matter. The customary method of testing such calculations is to apply them to the float and sink fractions of individual coals on which British thermal unit values have been determined. This presupposes that the pure coal substance in the float fractions has the same calorific equivalent as the pure coal substance in the sink fraction, an assumption that has yet to be demonstrated by experiment. However, following the procedure of former investigators, calorific equivalents have been calculated to a mineral-matter-free basis by the following formula:

$$\text{B.t.u. (mineral-matter-free)}\ \frac{\text{B.t.u.} - 50\text{S (pyritic)}}{100 - \text{M.M.}} \times 100$$

in which M.M. is determined from the ash analysis in the manner already described. The calculated B.t.u. are shown in Table 5.

In four out of the five cases, the agreement between float and sink mineral-matter-free calorific equivalents is better than that obtained by the Parr formula. One striking feature is the agreement between samples R and RW, which represent the same coal before and after cleaning, RW having been cleaned by a dry process. Furthermore, the agreement

TABLE 4.—*Detailed Calculation for Coal A—Original*

Step	Unit of Quantity	Constituent, Per Cent												Mineral	Mols	Mol. Wt.	Wt., Per Cent
		SiO_2	Al_2O_3	Fe_2O_3	TiO_2	P_2O_5	CaO	MgO	Na_2O	K_2O	SO_3	Pyritic S	Sulfate S				
1	Per cent in ash	40.4	38.6	4.7	1.5	4.0	5.6	0.5	0.4	0.9	2.6						
	Per cent in coal	2.87	2.74	0.334	0.107	0.284	0.398	0.0355	0.0284	0.0639	0.185	0.19	0.02				
2	Mols in coal	0.04759	0.02681	0.00209	0.00133	0.002	0.0071	0.00088	0.000462	0.000682	0.0023	0.00593	0.000625	Pyrite	0.00297	120	0.36
3 & 4	Mols subtracted			0.00149								0.00593					
	Mols remaining			0.00060													
5	Mols subtracted						0.00063						0.000625	Gypsum	0.00063	172	0.11
	Mols remaining						0.00647										
6 & 7	Mols subtracted					0.002	0.0065							Apatite	0.00130	505	0.65
	Mols remaining																
8	Mols subtracted						0.00							Calcite			
9	Mols subtracted	0.00088	0.00044	0.00044				0.00088						Prochlorite	0.00044	482	0.21
	Mols remaining	0.04671	0.02637	0.00016													
10	Mols subtracted			0.00016										Limonite	0.00008	374	0.03
	Mols remaining																
11	Mols subtracted				0.00133				Total 0.001144[a]					Rutile	0.00133	80	0.11
12	Mols subtracted	0.00687	0.00343						0.001144					Muscovite	0.00114	783	0.89
	Mols remaining	0.03984	0.02294														
13	Mols subtracted	0.03984	0.01992											Kaolin	0.0199	258	5.13
	Mols remaining		0.00302														
	Mols subtracted		0.00302											Diaspore	0.00302	120	0.36
																	7.85

[a] Sum of mols Na_2O and mols K_2O assumed to equal mols muscovite.

between the calorific equivalents, on the dry mineral-matter-free basis, for original float and sink portions is very close for each sample.

TABLE 5.—*Variation of B.t.u. of Pure Coal* CALCULATED IN SEVERAL WAYS

Sample	Ash-free	Difference between Float and Sink	Parr	Difference between Float and Sink	Fieldner and Selvig	Mineral-matter-free	Difference between Float and Sink
	B.t.u.		B.t.u.		Difference between Float and Sink[a]	B.t.u.	
A. Original	15630		15760			15740	
Float	15740		15800			15740	
Sink	14960	− 780	15470	−330	−270	15710	− 30
C. Original	15660		15860			15840	
Float	15790		15860			15850	
Sink	14890	− 900	15670	−190	+ 70	15720	−130
E. Original	15700		15820			15820	
Float	15750		15820			15810	
Sink	15160	− 590	15810	− 10	+180	15930	+120
R. Original	15580		15790			15840	
Float	15790		15850			15860	
Sink	14740	−1050	15500	−350	− 50	15820	− 40
RW. Original	15641		15810			15850	
Float	15790		15860			15850	
Sink	15000	− 790	15630	−230	0	15850	0

[a] Calorific equivalents not calculated. Differences taken from A. C. Fieldner, W. A. Selvig and F. H. Gibson: Application of Ash Corrections to Analyses of Various Coals. *Trans.* A.I.M.E. (1932) **101**, 226.

TABLE 6.—*Ratios of Mineral Matter to Ash*

Sample	M.M.-pyrite to Ash-Fe_2O_3 Equiv. to Pyrite	M.M. to Ash
A. Original	1.09	1.11
Float	1.01	1.01
Sink	1.13	1.15
C. Original	1.10	1.17
Float	1.12	1.15
Sink	1.13	1.19
E. Original	1.11	1.16
Float	1.12	1.17
Sink	1.12	1.19
R. Original	1.13	1.20
Float	1.12	1.13
Sink	1.16	1.24
RW. Original	1.14	1.19
Float	1.10	1.11
Sink	1.15	1.23

By subtracting the pyrite and iron oxide equivalent to pyrite from the mineral matter and ash respectively and dividing the former by the latter, it is possible to obtain a ratio that corresponds to the correction factor for converting ash to mineral matter. Table 6 gives such ratios as well as ratios of mineral matter to ash.

The magnitude of the corrections on the pyrite-free basis is in every instance but one larger than the factor of 1.08 as used in the Parr unit coal formula. However, the unit coal formula overcorrects for sulfur, since all of the sulfur is treated as though it were pyritic. Therefore it overcorrects for pyrite and undercorrects for the loss on ignition of other minerals. As the mineral matter approaches more nearly to 100 per cent pyrite the Parr formula becomes more nearly true.

As is to be expected, there is some variation in the correction factors for original, float and sink samples. This variation indicates the difficulty in arriving at a single correction formula that will be correct in all cases. Thus, following Parr's procedure, if we assume 10 per cent loss on ignition from the nonpyrite minerals, the noncoal formula becomes:

$$1.10\ \mathrm{A} + 0.5\ \mathrm{S}$$

For each increase of 0.02 A the factor for sulfur must be diminished by $\frac{1}{40}$, so we arrive at:

$$\begin{aligned} &1.12\ \mathrm{A} + \tfrac{19}{40}\ \mathrm{S} \\ &1.14\ \mathrm{A} + \tfrac{18}{40}\ \mathrm{S} \\ &1.16\ \mathrm{A} + \tfrac{17}{40}\ \mathrm{S}, \text{ etc.} \end{aligned}$$

The difficulty with such calculations is that no one formula can be expected to apply always with the same accuracy. For this reason, in coal classification it seems wiser to use the simplest formula that will give approximately uniform results for most cases. Exceptions can then be subjected to more detailed investigation. The producer is in urgent need of systematic coal classification, but it is useless to expect him to utilize elaborate correction factors, just as it is useless to expect him to sell his coal on the basis of a seam classification when he puts it through an elaborate cleaning and blending process and knows that the coal that goes into the car is not the same as the coal in the seam. The important point now is to arrive at the best possible system of scientific classification as quickly as possible, to keep it as simple as possible, and then to allow the coal industry to use as much of it as is practicable.

If one calculates the weights of the respective minerals in the float and sink samples certain significant trends may be observed. Gypsum, for example, tends to remain in the float whereas muscovite, kaolin and limonite tend to go to the sink. Table 7 illustrates this.

The authors fully realize the shortcomings of the procedure discussed herein. For example, the chlorites represent a group of minerals of variable composition. However, the effect of a somewhat different composition would be relatively small so long as the amount of chlorites was small. Incidentally, the correction factor for chlorite of the formula accepted is 1.08, which is somewhat smaller than the 10 per cent loss obtained by Selvig and Pohle[5] on heating in the volatile furnace for seven minutes.

Another point that should be emphasized is that kaolin is calculated by assuming that as much of it is present as could be formed by combining SiO_2 and Al_2O_3 in the ratio 2 to 1 after the amount of these components necessary to form all other aluminum silicates has been deducted from the amounts originally shown in the ash analysis. Consequently, an excess of one or the other, but not both, is shown. On the other hand, free quartz was found in every sample and it is likely that both quartz and diaspore may exist in some samples. This will explain the case of R and RW where diaspore appears in the float fractions but not in the original coal. Since kaolinite contains two molecules of water of crystallization whereas diaspore contains only one, this assumption makes some difference in the total mineral matter. Thus, if the kaolin in the sample A original were all quartz and diaspore, we would have 2.39 per cent quartz plus 2.39 per cent diaspore, making a total of 4.78 instead of 5.13. On the other hand, aluminum oxide exists in greater degrees of hydration; as, for example, in halloysite. Furthermore, it is highly improbable, both from the geological and thermochemical viewpoints, that diaspore and quartz will exist to anywhere near the exclusion of kaolin minerals. As a matter of fact, diaspore was observed in only two of the samples examined.

TABLE 7.—*Calculated Minerals in Float and Sink Fractions*

Mineral	Sample A		Sample C		Sample E		Sample R		Sample RW	
	Float	Sink	Float	Sink	Float	Sink	Float	Sink	Float	Sink
Pyrite	0.06	0.40	0.26	1.51	0.17	0.90	0.04	2.55	0.10	1.71
Gypsum	0.02	0.02	0.04	0.06	0.05		0.04	0.19	0.04	0.09
Apatite	0.03	0.43	0.01	0.02	0.01	0.01	0.03	0.04	0.03	0.03
Calcite	0.04		0.18		0.13	0.02	0.13		0.14	
Prochlorite	0.11	0.09	0.17	0.10	0.04	0.04	0.06	0.04	0.07	0.06
Limonite		0.08		0.16		0.12	0.04	0.52	0.06	0.23
Rutile	0.05	0.05	0.05	0.06	0.05	0.04	0.05	0.07	0.06	0.05
Muscovite	0.30	0.54	0.29	0.77	0.33	0.35	0.70	0.81	0.48	0.55
Kaolin	1.50	3.98	1.69	3.21	1.66	2.13	1.63	3.90	2.08	3.21
Quartz			0.12	0.60	0.06	0.23		0.31		0.22
Diaspore	0.25	0.12					0.04		0.06	

In spite of its shortcomings, the authors submit that the general point of view adopted in this paper constitutes the only logical method in attempting to arrive at some knowledge of pure coal on the basis of experimental data. It is not suggested that the minerals listed in Table 1 are all the minerals in the coal samples studied. They have, however, been definitely identified, consequently furnish a basis for the rationalization of the chemical analysis. The subjected is being investigated further in this laboratory.

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[5] W. A. Selvig and W. D. Pohle: Losses of Volatile Matters in Coal by Standard Method. *Ind. & Eng. Chem.* Anal. Ed. (1933) **5**, 239–241.

ACKNOWLEDGMENTS

The authors wish to acknowledge with gratitude the assistance received from Glen Burgener, who made the petrographic examinations; G. A. Brady, who did the analytical work, and O. P. Brysch, who made some of the float and sink separations.

DISCUSSION

(*A. C. Callen presiding*)

G. THIESSEN,* Urbana, Ill.—Dr. Gauger, with his associates, has arrived at much the same conclusions as I have, by approaching the subject through different viewpoints.

R. D. HALL,* New York, N. Y.—Has barium been discovered in coal ash? After some carboniferous shales had been burned with coal, witherite appeared as a white coating on some of the brick, proving that barium was present in the shale. It might be expected, therefore, in shaly constituents of coal. Germanium also might be found; it has been reported as present in German and English coal ash.

F. W. Clark[6] says that analyses of shales suggest the presence of 30 per cent of feldspar, and similar analyses of sand, the presence of 11.5 per cent of the same mineral. One would expect to find some confirmation of this statement if an investigation were made of shales and sandstone, if not of coal ash.

Our clays need examination; there may be in them aluminum compounds, not only feldspar, but diaspore, gibbsite, bauxite and alunite. Many shales and shaly clays in the coal formations are said by James Dewar, in the Encyclopædia Britannica, Ninth Edition, to yield alum when acted on by the atmosphere. Alum is manufactured from such material at Campsie, and other places near Glasgow. Alum shales are found in the Lias at Whitby in Yorkshire and are similarly used. Other like beds are found in the brown-coal Tertiary formations of Germany. Freshly dug they show not a trace of alum, but the mineral is formed after exposure to the air and when the pyrite is decomposed by heat. Near Glasgow, clay and shale are taken from old coal pits, where it has lain in moist air for many years at a temperature of 62°. Efflorescent salts appear, which prove to be mixtures of sulfates of iron and sulfates of alumina. The shale is leached and left exposed to the elements, the rain water drawn from it being collected. Then the liquid is boiled in stone boilers till it reaches a sufficient stage of concentration. After the temperature is raised, sulfate of potash or muriate of potash is added, which converts the sulfate of alumina to alum. Alum crystallizes first, and thus it can be separated from any iron sulfate present.

Nothing is said in the last edition of the Encyclopædia about these processes, so they may have been discontinued, but they suggest possibilities of recovering alum and sulfate of alumina from coal clays and shales. Both have commercial uses, and the sulfate might profitably be converted to aluminum, that being one means of making that metal.

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* Engineering Editor, *Coal Age*.

[6] F. W. Clark: Data of Geochemistry, 31. 1911.

G. Thiessen.—What are the proportions of pyrite, clay and calcite?

A. W. Gauger.—There is little calcite in Central Pennsylvania. Examination of three Miller seam coals which had in general the same minerals showed their ash fusion temperatures (softening point) to range from 2100° to 3000° F.

H. H. Lowry,* Pittsburgh, Pa.—Not all of the ash comes from inorganic matter in the coal, since vacuum tar and benzene extract both leave an inorganic residue on combustion.

Member.—Vitrain in some English coals has as little as 0.5 per cent ash, which is unique in that it is organic ash.

A. C. Fieldner,† Washington, D. C.—Are manganese minerals reported?

A. W. Gauger.—They were not observed although both manganese and barium minerals may be present in small quantities. The spectroscope reveals almost everything in the periodic table.

G. H. Cady,‡ Urbana, Ill.—The Illinois State Geological Survey has examined columns of Illinois coal from top to bottom, using float-and-sink separation after fine grinding. This has yielded a detailed record of the mineral constituents of the coal. Four minerals, pyrite, kaolinite, related clay minerals and calcite make up 96 to 98 per cent of the total mineral recovered. Feldspars, quartz, micas, tourmaline, epidote and other resistant minerals compose the remainder. The latter probably have little effect, if any, on the fusion point of the ash. Many of the facings in Illinois coal are not calcite but kaolinite. Those coals low in pyrite are high in kaolinite and have high ash fusion temperatures.

W. A. Selvig,§ Pittsburgh, Pa.—A paper was published by the Fuel Research Board of Great Britain[7] which is of value to anyone interested in the chemical elements in coal ash. It contains an excellent bibliography.

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† Chief Engineer, Engineering Experiments Division, U. S. Bureau of Mines.

‡ Senior Geologist, Illinois State Geological Survey.

§ Chemist, U. S. Bureau of Mines.

[7] Methods for the Quantitative Analysis of Coal Ash. Physical and Chemical Survey *Pub.* 28, Fuel Research Board of Great Britain.

Effect of Oven Humidity on Accelerated Weathering Tests of Coal

By E. Stansfield* and K. C. Gilbart,* Edmonton, Alberta

(New York Meeting, February, 1934)

This test was described in a privately circulated First Progress Report on Slacking Characteristics of Coal, by A. C. Fieldner and W. A. Selvig, May 1, 1928. The test was described in detail in U. S. Bureau of Mines *Report of Investigation* 3055, in November, 1930.[1]

The test, in brief, is conducted as follows: A sample of freshly mined coal consisting of 18 to 25 pieces approximating 1¼ cubes, and weighing 500 to 600 grams, is taken for the test. The sample is subjected to a "blank sieving" of carefully regulated degree, on an 8-in. circular sieve with 0.263-in. square openings, and the undersize and oversize weighed. The oversize is placed on a sheet-iron pan and dried for about 24 hr. in an A.S.T.M. air-drying oven maintained at 30° to 35° C. through which a current of warm air is drawn. The sample is then cooled to room temperature and immersed for one hour in water at the same temperature. The water is then drained off, the sample dried for 24 hr. as before, cooled, sieved as before, and the oversize and undersize weighed. The percentage weight of the undersize is calculated and reported as the first cycle loss. This loss, corrected by subtracting the percentage loss in the blank sieving, is termed the slacking index. The sample may be subjected to further cycles of wetting, draining, air-drying, sieving and weighing.

In a letter (August, 1932) W. A. Selvig approved making the blank sieving on a duplicate sample, rather than on the sample used for the test, where sufficient coal was available. This method is now employed in these laboratories.

The accelerated weathering test has been employed in the laboratories of the Research Council of Alberta, at Edmonton, since 1928, but not found entirely satisfactory as repeat determinations with samples taken from the same or adjacent mines frequently showed serious differences. Differences must be expected as the coal may vary in the seam, and the selection of a typical set of lumps approximating 1¼-in. cubes

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* Fuels Division, Research Council of Alberta.

[1] A. C. Fieldner, W. A. Selvig and W. H. Frederic: Accelerated Laboratory Test for Determination of Slacking Characteristics of Coal. U. S. Bur. Mines *Rept. of Investigations* 3055 (1930).

is always difficult; nevertheless the test plays an important part in the evaluation of coal and it seemed advisable to locate and reduce the cause or causes of the serious differences noted above.

Checking or slacking of coal during weathering is characteristic of high-moisture coals, and is caused by the shrinkage of the coal when it loses moisture. The extent of the slacking is related to the extent of the moisture loss. The rate of drying of coal and the extent to which it dries when exposed to air are closely related to the humidity of the air. In the test, as described above, there are no limits prescribed for the humidity of the air circulated through the drying oven, and in our opinion this is the main *controllable* source of error in the test.

If air of 10 per cent relative humidity drawn from a room at 20° C. (frequent winter conditions in Edmonton) is heated to 30° C. in the oven, the resulting humidity of the air reaching the coal is about 5 per cent; while on a hot, humid day the laboratory air might have a humidity of 80 per cent, and a temperature of 30° C., and thus reach the coal with a humidity of 80 per cent. Variations of such a range of humidity will, as shown below, entirely prevent uniformity of result.

A preliminary series of experiments was made with air of partially controlled humidity circulated as usual through an A.S.T.M. air-drying oven. The air for the oven was drawn down through a 16-ft. tower in which a fine mist of water could be sprayed.

In one run made without the spray the heated air leaving the oven had a relative humidity of 3 per cent. The coal, which contained 24 per cent moisture as tested, lost 12 per cent in the air-drying and showed a slacking index of 74. In a second run with the spray the air left the oven with a humidity of 40 per cent, the air-dry loss was now only 7 per cent and the slacking index 34. This confirmed the suspicion that a controlled humidity is necessary for this test.

The use of a tower in which to moisten the inlet air is cumbersome, not suited to exact control, and not likely to be adopted as regular equipment. A simpler method was therefore sought. It was finally decided to change from a continuous stream of air through the oven to a rapid circulation of air within a closed chamber with control of temperature and humidity. This system permits the use of saturated salt solutions, with excess of free salt, for humidity control.

The air-drier chamber constructed is shown in Fig. 1. It consists of an iron box, with welded seams, 31 by 23 by 19 in. inside dimensions. The lid, also of welded iron, fits down into a liquid seal in a small trough, which is an integral part of the box. The lid and seal are shown in the sectional elevations, but not in the plan. The iron box is enclosed in an outer wooden box for heat insulation.

Air circulation is provided by a 12-in. desk fan, and the air is compelled to make a complete circuit of the box by a metal partition down the middle, as shown in the plan. The fan blades rotate within an opening in a cross diaphragm. The motor is surrounded by a water-cooled, copper coil to absorb the heat it generates. The air leaving the fan passes over two small carbon filament lamps, which are connected to a thermoregulator and suitable cutoff set to control the air temperature to 30° C. This air passes over and through a stack of four graniteware trays, 15 by 10 by ¾ in., containing the desired salt and solution. The tempered air passes the thermoregulator at the end of the box and returns over the trays of coal on the other side of the partition. Racks for eight small trays are provided with capacity to dry two samples at the same time. Openings are provided in the lid for wet-bulb and dry-bulb thermometers for determination of relative humidity; but in the earlier work a recording hair hygrometer was kept in the box to provide a full record of humidity conditions. The air humidity reaches approximate equilibrium within 2 hr. of the start, even when wetted coal is being air-dried.

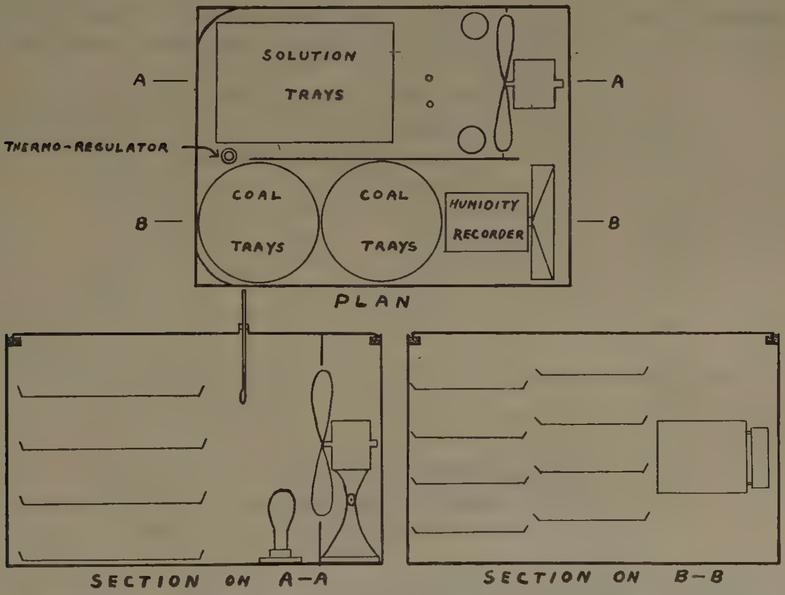

FIG. 1.—OUTLINE OF APPARATUS FOR DRYING COAL AT CONTROLLED HUMIDITY AND TEMPERATURE.

Either oil, or a glycerin-water mixture, can be used in the seal for the lid. Two oil-sealed tubes were provided to admit or discharge air as required by change of barometric pressure.

The salt and its solution are spread out on the shallow trays to expose a large surface to the air. Free salt must be exposed above the solution, otherwise the top layer may absorb water and become less than saturated, and, being lighter than the saturated solution, remain on the top and out of contact with the solid salt below. Two ways have been employed to keep the solid salt exposed. One is to divide each tray into four parts by lengths of ¾-in. glass rods or tubes, and to keep the salt in two parts. Another way is to place muslin bags of the salt in each tray. Enough salt must be provided to absorb all the water from a wet sample, and enough tray capacity so that the trays do not overflow. It is a very simple matter to re-dry the trays at the end of a run by exposure in the laboratory or in a current of warm air.

Unless the box is tightly closed, dry air from the laboratory is likely to enter and cause such evaporation that the solution "freezes" over and humidity control is at once lost. This difficulty was overcome by the use of the airtight apparatus described above. It might be overcome by the use of mechanical stirrers in the solution trays.

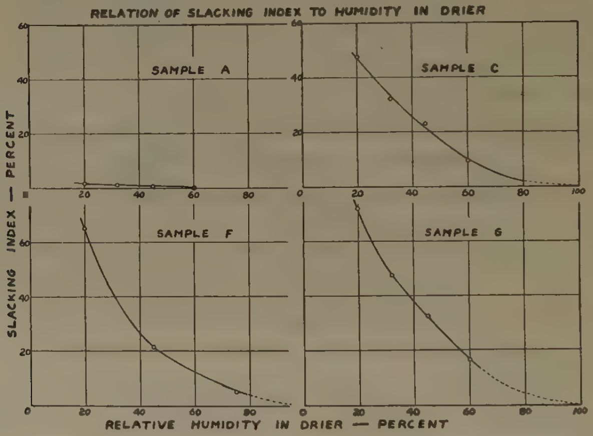

FIG. 2.—EFFECT OF HUMIDITY IN THE DRYING OVEN.

Five salts were used to get a range of relative humidities from 20 to 75 per cent at 30° C. potassium acetate for 20 per cent humidity, magnesium chloride 32 per cent, calcium nitrate 45 per cent, ammonium nitrate 60 per cent and sodium chloride 75 per cent.

Tests with controlled humidity were made with three distinct ranks of coal as described below; these were nonslacking, slightly slacking and slacking coals respectively.

Three high-rank, coking, bituminous coals were tested. Air-dried at 20 per cent humidity the slacking indices did not exceed one per cent. At higher humidities the values would be even less and were not determined. Humidity control, therefore, is not essential for such coals.

Two noncoking coals tested were border-line coals between bituminous and subbituminous. In earlier work with uncontrolled humidities and such coals slacking indices as high as 9 had been found. These coals, A and B in Tables 1 and 2, showed slacking indices that were below 2 even at 20 per cent humidity, which suggests that some of the earlier tests must have been made with far drier air in the oven. The air-dry loss as shown in Table 1 and the slacking index in Table 2 decrease regularly with increase of humidity in the air-drying oven. This is shown more clearly for coal A in the curve of Fig. 2. Some humidity control is clearly advisable when testing such coals.

TABLE 1.—*Influence of Humidity in Drier on Drying Loss*

Sample	Typical Analysis				Air-drying Loss, Per Cent				
					Relative Humidity in Drier, Per Cent				
	Moisture, Per Cent	Ash, Per Cent	Vol. Matter, Per Cent	Cal. Value, B.t.u. per Lb.	75	60	45	32	20
A	8	8	36	11,400		1.1	1.5	1.7	2.3
B	10	6	36	11,700				2.1	2.5
C	20	7	27	9,500		3.7	5.1	6.9	8.4
D	21	8	28	8,900			6.8	9.7	11.6
E	25	5	28	9,100		4.7	6.7	8.6	10.4
F	25	9	27	8,400	1.3		6.9		11.1
G	27	5	29	8,600		8.5	11.5	14.3	15.8

TABLE 2.—*Influence of Humidity in Drier on Slacking Index*

Sample	Slacking Index				
	Relative Humidity in Drier, Per Cent				
	75	60	45	32	20
A		0.1	0.3	1.2	1.7
B				0.6	1.3
C		9.3	23.3	31.9	47.8
D			28.7	41.0	51.0
E		16.5	19.1	30.6	49.7
F	5.1		21.7		65.5
G		16.7	32.9	47.5	71.8

Five lower rank slacking coals have also been tested, and these results (coals C to G) are also shown in the tables and by curves. The changes of slacking index with humidity are here seen to be very large, a reduction from 40 to 20 per cent humidity on the average results in a doubling of the slacking index. Close humidity control is therefore essential with such coals.

In conclusion, it is submitted that the accelerated weathering test should be amended to include a specification for the humidity in the air-drying oven.

This work is being published at the suggestion of A. C. Fieldner and W. A. Selvig, who have taken a sympathetic interest in its progress. It would seem that a humidity value selected from within the range of 20 to 40 per cent would best agree with the large number of slacking indices already determined in the laboratories of the U. S. Bureau of Mines.

The National Research Council of Canada has given financial assistance in this work.

DISCUSSION

(*A. C. Fieldner presiding*)

E. Stansfield.—It may be possible to evaluate coals by their moisture-losing properties, although early work by Fieldner and Selvig indicated that slacking was not proportional to moisture loss. The retention of moisture by coal depends on the relative humidity of the surrounding atmosphere. Tests to date indicate that nonslacking coals are characterized by not losing more than 1.7 per cent moisture when the humidity of the atmosphere falls from 97 to 60 per cent. On the contrary, one slacking coal was found to lose only 1.5 per cent, under these conditions, but the next lowest value was 2 per cent.

The apparatus described is now in use at Edmonton and Ottawa, and the U. S. Bureau of Mines expects to have one in commission shortly. It is possible to dry as much as 150 grams of coal, to definite humidity, over a suitable salt in an evacuated desiccator; this may prove a more drastic method of air-drying than the one described, but the equipment would be readily available in any laboratory. This method will be tested.

A. C. Fieldner,* Washington, D. C.—There is urgent need of the standardization and adoption of such a test, as it is needed in distinguishing between bituminous and subbituminous coals.

W. A. Selvig,† Pittsburgh, Pa.—The U. S. Bureau of Mines has built a constant-humidity oven, but has not yet used it. The relative humidity ordinarily prevailing in the ordinary air-drying oven at the Pittsburgh station of the Bureau of Mines averages about 20 per cent in winter and 40 per cent in summer. In a year's record the range was from 10 to 55 per cent.

Member.—In what area does the nonslacking coal occur that lost only 1.5 per cent, and how does the constitution of that coal differ from the others?

E. Stansfield.—This is a dense coal of the highest rank in the Edmonton horizon; 15 per cent moisture, typically humic.

H. G. Turner,‡ State College, Pa.—The vacuum dessicator is an attractive suggestion but might not some volatile matter be lost during the test? It might be advisable first to dry in an oven, then in the desiccator.

E. Stansfield.—The only volatile matter to be lost in the dessicator would be occluded methane.

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[*] Chief Engineer, Experiment Stations Division, U. S. Bureau of Mines.

[†] Chemist, U. S. Bureau of Mines.

[‡] Director of Research, Anthracite Institute.

Oxidation of Coal and the Relation to Its Analysis

By E. Stansfield,* W. A. Lang* and K. C. Gilbart,* Edmonton, Alberta

(New York Meeting, February, 1934)

It has long been known that coal is unstable and oxidizes in air, even at ordinary atmospheric temperatures; also, that such oxidation affects the analysis of coal. Nevertheless little or no precaution is taken to prevent oxidation in the ordinary methods of analysis, and the possibility of error due to oxidation is sometimes ignored even in researches on coal.

Although the general nature of coal oxidation is well known, and many workers have made careful studies of the rate of oxidation of different types of coal exposed to oxygen at different temperatures, yet the amount of oxidation coals undergo in the ordinary process of analysis, and its influence on the determined analysis, is not well known, and the published data are widely scattered through the literature. This paper is a collection, from this laboratory, of data that bear on the subject, together with a brief summary of three methods employed here to evaluate the oxidizability of coal.

The post-Carboniferous coals of Alberta range all the way from anthracite down to lignite. As some of them are notably susceptible to oxidation, the necessity for care was early forced upon our attention, and oxidation has been reduced either by the removal of air with a vacuum pump or by the displacement of air with natural gas (methane).

The two routine analytical operations most liable to involve oxidation are the preliminary, partial drying of the crushed coal (called air-drying) and the determination of moisture. The first is a prolonged operation on coarse coal at ordinary temperatures, and the second a short treatment on powdered coal at higher temperatures. Oxidation also occurs in coal samples stored in ordinary sample bottles.

In order to gain information for this study, determinations were recently carried out on the same samples in air, in natural gas, and in vacuo. Sixteen coals were tested; three would be classed in the United States as coking bituminous coals, two as noncoking, nonslacking, bituminous coals, and the rest as subbituminous. Table 1 gives typical analyses of these coals as mined.

Manuscript received at the office of the Institute Dec. 1, 1933.

* Fuels Division, Research Council of Alberta.

ABSORPTION OF NATURAL GAS BY COAL

The statement has been made that natural gas is absorbed by coal and should not be used in work on coal. Edmonton natural gas is mainly methane, with only 3 per cent ethane and propane, and is less likely to be absorbed than a richer gas. Repeated experiments showed such close agreement between tests with coal dried in vacuo and with coal dried in gas that the gas has been used with confidence. It is by far the cheapest and simplest method available here for displacing air. The following recent work, however, has cast doubt on the safety of exposing coal to natural gas at room temperature, and further studies will be made.

TABLE 1.—*Typical Analyses of Coals Tested, As-mined Basis*

Coal	Moisture, Per Cent	Ash, Per Cent	Vol. Matter, Per Cent	Fixed Carbon, Per Cent	Calorific Value, B.t.u. Per Lb.
Group A. Bituminous, coking (bituminous)[a]					
A-1	3	12	20	65	13,200
A-2	2	9	28	61	13,700
A-3	3	9	30	58	13,700
Group B. Bituminous, noncoking, nonslacking (subbituminous)					
B-1	10	6	36	48	11,700
B-2	11	14	33	42	10,100
Group C. Subbituminous, noncoking, slacking (domestic or lignite)					
C-1	13	8	34	45	10,600
C-2	15	10	22	53	10,100
C-3	17	10	30	43	9,700
C-4	20	7	27	46	9,500
C-5	21	8	28	43	8,900
C-6	21	9	28	42	8,800
C-7	25	5	28	42	9,100
C-8	25	7	28	40	8,700
C-9	25	9	27	39	8,400
C-10	25	11	26	38	8,100
C-11	28	4	28	40	8,600

[a] Present Canadian classification given in brackets.

A number of one-gram samples of each of four coals were dried 2 hr. at 105° C. in a Freas vacuum oven. Some samples of this dried coal were reheated 1 hr. as before; others were reheated in a current of hot, dry, natural gas in a toluene oven at 106° C.; and others given similar treatment in hot, dry air. After each heating the samples were cooled in an exhausted desiccator and weighed. Another batch of dried samples was exposed to natural gas for 16 hr. at room temperatures; then weighed at once, reweighed after ½ hr. in an exhausted desiccator, and reweighed after 1 hr. repeat heating in the vacuum oven. The results are shown in Table 2. It can be seen that the change in weight caused by heating in gas was barely outside experimental errors, and notably less than the change due to heating in air. The coal exposed 16 hr. to gas in the cold, on the contrary, showed distinct absorption. The absorbed gas was not removed by evacuating in the cold, and only three-quarters of it was removed by heating 1 hr. in vacuo.

A preliminary experiment was made in which a weighed sample of pulverized coal, C-7, was exposed 6 hours to dry, natural gas and then burned in the calorimeter. The determined calorific value was about 100 B.t.u. higher than with a similar sample not exposed to gas. The absorption of 1 per cent of methane would raise the calorific value of a 10,000 B.t.u. coal by about 140 B.t.u.

It should be noted that in the above tests powdered coal was used, while air-drying is with crushed coal that doubtless absorbs gas less rapidly. Coal for analysis in this laboratory is ball-mill ground and normally will pass a 100-mesh screen. The effect on the calorific value of air-drying crushed coal in gas will be further discussed in connection with Table 4.

TABLE 2.—*Percentage Changes of Weight in Ground Coals Dried* 2 *Hr. in a Vacuum Oven at* 105° *C. and Retreated as Below*

Coal	Reheated 1 Hr. in Vacuum Oven 105° C.	Reheated 1 Hr. in Gas in a Toluene Oven 106° C.	Reheated 1 Hr. in Air in a Toluene Oven 106° C.	Exposed 16 Hr. to Dry Gas at Room Temperature		
				Weighed after Exposure	Weighed after ½ Hr. in a Vacuum Desiccator	Weighed after Reheating 1 Hr. in Vacuum Oven at 105° C.
A-2	−0.01	+0.10	+0.11	+0.55	+0.53	+0.11
B-1	+0.02	+0.12	+0.27	+0.79	+0.76	+0.24
C-1	−0.06	−0.07	+0.10	+0.68	+0.68	+0.15
C-7	−0.01	+0.06	+0.24	+1.15	+1.12	+0.30
Average..	−0.01	+0.05	+0.18	+0.79	+0.77	+0.20

Nature of Coal Oxidation

Stopes and Wheeler[1] stated this clearly in 1923. The absorption of oxygen by newly won bituminous coal is initially very rapid, even at ordinary atmospheric temperatures, but the rapid absorption soon gives place to a slow but long continued absorption. The initial rapid absorption at low temperatures does not appear to be accompanied by the formation of water or oxides of carbon, but these products of combustion appear during the slow phase. Stopes and Wheeler suggested that oxygen is first added to the coal, then, notably at higher temperatures, products of combustion split off. More oxygen can then be added, and so on, the coal after the splitting off being little altered from the original coal. This refers to bituminous coal. Lower rank coals are more oxidizable, and split off products of combustion more readily, but the processes involved appear similar.

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[1] M. C. Stopes and R. V. Wheeler: The Spontaneous Combustion of Coal in Relation to Its Composition and Structure. *Fuel* (1923) 29.

Although the oxidized coal may show little change in its susceptibility to further oxidation, etc., it is well known that it is changed in other ways, notably in its solubility in caustic potash, as insoluble ulmins become soluble.

The weight and other changes involved in oxidation during coal storage and analysis can be explained in general by this summary of the processes of oxidation.

Oxidation and the Storage of Coal Samples

If samples of coal are stored, after analysis, in a dry room in bottles with ground glass stoppers and the coal then reanalyzed, the second analysis normally shows a higher moisture unless the stopper is a misfit. The calorific value, as determined, will show a distinct drop; but if this value is calculated to the dry-coal basis the drop is often negligible. The coal presumably takes up oxygen with a gain in weight, and a drop in calorific value per unit weight. In the second moisture determination not only the original moisture but also some water and other products of oxidation are driven off and weighed (by difference). The remaining dried coal has a heat value little different from that of the original coal.

A series of 19 air-dried coals were retested for moisture after storage for 12 to 36 days. The moisture increase varied from 0.08 per cent on a bituminous coal with 0.5 per cent of water, stored 33 days, to 1.3 per cent on a coal with 7 per cent of water stored 34 days. The moisture formation during storage is slight in high-rank coals and large in low-rank coals.

The increase of moisture is at least mainly due to oxidation, and not to decomposition, of the coal. A moist coal, similar to C-8 in Table 1, between 8 and 14 mesh in size, was exposed for a month at laboratory temperature to a stream of dry air. The weight and moisture content of the coal were determined before and after the experiment, and also the moisture carried off in the air stream. The moisture in the original coal was 26.35 per cent, and the moisture in the coal at the end, plus the moisture collected from the air leaving the coal, was 29.25 per cent, a moisture increase of 2.9 per cent. In a parallel experiment with natural gas instead of air the moisture increase was only 0.14 per cent, a clear proof that oxygen was required for the formation of water.

There was an increase of 0.62 per cent in the weight of dry coal substance in the experiment in air, and an increase of 0.46 per cent in gas. The latter gain may have been due to absorbed gas.

OXIDATION OF COAL DURING AIR-DRYING[2]

Coal during air-drying loses weight as it dries and gains in weight as it oxidizes. The net loss of weight is thus less than the true moisture loss. As this drying is only partial it is necessary in comparison of methods to grind the air-dried material, determine the residual moisture and compute the total moisture. The air-dried and oxidized coal will, however, lose not only its residual moisture but also some of its water of oxidation during the moisture determination at 106° C. The net result of oxidation during drying on the determined total moisture may therefore be slight, if the gain in weight during oxidation is balanced by the subsequent loss of water of oxidation. For the same reason the determined calorific value may be but little affected.

Prior to 1931 coals were air-dried here for 48 hr. in a current of air of 60 per cent humidity at room temperatures. In that year the air drying was changed to natural gas of 60 per cent humidity at 30° C., in order to reduce oxidation. Some time later a compilation of analyses was made on a dry coal, common ash basis, for all mines from which samples had been analyzed before and after the change. It was found that for all coals the calorific values of samples air-dried in gas averaged 60 B.t.u. higher than those of samples dried in air. The more recent coals (Edmonton geological horizon) showed an average difference of 90 B.t.u.

These increases in calorific value might be due to reduced oxidation, to absorption of natural gas, or to both. The former appeared to be at least the main cause, and in later work when air was used the time was reduced from 48 to 24 hours.

Tables 3 and 4 show the results of experiments to determine the effect of air-drying in air and in gas on total moisture and on calorific value respectively. Some coals for comparison were air-dried in a highly evacuated desiccator with controlled humidity at 30° C., and others in a Freas vacuum oven at 50° C. Air leaks around the door made it impossible to keep the absolute pressure in the oven below 2 in. of mercury. Vacuum drying, with only one exception, gave the higher total moisture, with an average difference of 0.25 per cent. Samples air-dried in air agreed fairly well as to total moisture with the samples treated in gas. The effect of air-drying on calorific value was less consistent. The average for all samples dried in gas was 20 B.t.u. higher than those dried in vacuum, and 70 B.t.u. higher than those dried in air.

There was, on the whole, little advantage shown for air-drying in gas. Vacuum drying at about 50° C. has the advantage of speed, but it is difficult to judge the time and temperature required to give an air-dried sample that will not rapidly lose or gain moisture during analysis. Air-drying appears to require further consideration.

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[2] As the partial drying of crushed coal in preparation for grinding and analysis is always known as "air-drying," this term is retained for clarity throughout this paper even when air is not employed.

Oxidation during Moisture Determination

A series of coals was tested for moisture in a toluene oven at 106° C. in a current of dry air and also in a current of dry natural gas, 3½ cu. ft. per hour in each case. In the Siau oven employed the air or gas is raised to the oven temperature before it enters the drying chamber. A comparison with a vacuum method was not made in this series because earlier work had shown excellent checks with gas, and coal has been shown not to notably absorb gas under the conditions of the test. Table 5 shows the results. In every case, as is natural, the determination made in air showed the lower moisture. The difference varied from 0.1 per cent with a bituminous coal to 0.6 per cent with a lower rank coal. The average difference was 0.3 per cent.

Table 3.—*Total Moisture Found with Different Methods of Air-drying*[a]

	Method of Air-drying			Differences	
Coal	(1) Vacuum Oven, 1–2 Hr. About 50° C., Per Cent	(2) Natural Gas, 24 Hr. at 30° C., Per Cent	(3) Air, 24 Hr. Room Temperature, Per Cent	1–2	1–3
A-1		2.55	2.55		
A-2	2.52	2.07	2.25	−0.45	−0.27
A-3	2.83	2.56	2.66	−0.27	−0.17
C-1	13.88	13.18	13.26	−0.70	−0.62
C-2	15.39	14.98	14.96	−0.41	−0.43
C-3	17.52	17.40	17.35	−0.12	−0.17
C-4	20.07	19.82	19.91	−0.25	−0.16
C-5	22.59	22.58	22.68	−0.01	+0.09
C-7		25.03	25.25		
C-8	26.48	26.45	26.29	−0.03	−0.19
Average...				−0.28	−0.24

[a] For 15 coals tested the total moisture when air was used for air-drying exceeded the value when gas was used by an average value of only 0.07 per cent.

It is submitted that reasonable accuracy with this determination requires the absence of air. The A.S.T.M. specifications[2] definitely require a current of air, but the British Engineering Standards Association[3] advocates a neutral gas or reduced pressure for oxidizable coals.

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[2] Standard Methods of Laboratory Sampling and Analysis of Coal and Coke. D 271, 29.

[3] British Standard Specification for the Sampling and Analysis of Coal. B.S.S. 404 (1930) and B.S.S. 420 (1931).

TABLE 4.—*Effect of Method of Air-drying on Calorific Value of Coal*[a]

Differences in B.t.u. per Pound from Value Determined on Vacuum-dried Coal

Coal	Dried in Gas 24 or 48 Hr. 30° C.	Dried in Air		
		12 Hr., Room Temperature	24 Hr., Room Temperature	48 Hr., Room Temperature
Comparison with coals dried 48 hr. in evacuated desiccator at 30° C.				
B-2	+ 30	−20		− 10
C-6	0	−20		− 50
C-9	+ 50	−70		−140
C-10	+ 50	−50		−110
C-11	0			−180
Comparison with coals dried 1 to 2 hr. in Freas vacuum oven at about 50° C.				
A-2	+ 50		+ 20	
A-3	− 10		0	
C-1	+100		+ 60	
C-2	+ 60		+ 10	
C-3	− 30		− 90	
C-4	+ 10		− 70	
C-5	+ 10		− 70	
C-7	− 30		−100	
C-8	+ 20		− 30	
Average..........	+ 20	−40	− 30	−100

[a] The average calorific value of all coals air-dried in gas exceeded that for coals air-dried in air by 70 B.t.u. per pound.

TABLE 5.—*Moisture Determinations on Air-dried Coal*

Coal	Heated in Toluene Oven at 106° C.		
	In Gas, Per Cent	In Air, Per Cent	Difference, Per Cent
A-1	0.79	0.71	0.08
A-2	0.73	0.63	0.10
A-3	0.86	0.68	0.18
C-1	8.34	7.75	0.59
C-2	8.31	8.08	0.23
C-3	12.23	11.91	0.32
C-4	10.95	10.47	0.48
C-5	13.39	13.07	0.32
C-7	12.89	12.44	0.45
C-8	16.95	16.61	0.34
Average.............			0.31

COAL OXIDATION AND CALORIFIC VALUE

The change in calorific value of coal during oxidation is somewhat difficult to follow since change of weight and of calorific value occur simultaneously, and it is frequently impossible to relate the final weight of oxidized coal to the original weight of fresh coal. Parr and Milner[4] say:

> There seems to be ground for the conclusion that oxygen is taken into the coal texture primarily in an adsorbed or surface condensation form. In this stage its presence is indicated by an increase in weight of the mass, and the loss in heat units corresponds to this weight increment; hence the heat loss is apparent and not real. But at slightly elevated temperatures the adsorbed oxygen goes over into actual chemical combination, with the evolution of water and carbon dioxide. Here there is an actual loss of heat, practically equivalent in amount to the heat of combustion represented by the weight of carbon and hydrogen in the water and carbon dioxide evolved.

Changes of heat value that were more apparent than real were discussed under storage of samples. Real changes were further studied by more extreme oxidation than occurs during analysis. Four portions of a pulverized coal sample were weighed out into the capsules used in calorimetric determinations. Two of these were burned at once in the calorimeter, and two were burned after oxidizing for 6 hr. in a current of dry air at 106° C. The change of weight during this drying was noted.

One coal, B-1, gave a calorific value of 12,020 B.t.u. as tested without oxidation; and 12,000 B.t.u. on the weight taken when oxidized between weighing and burning, a drop of only 20 B.t.u.

The original coal contained 7.1 per cent moisture, but the net loss of weight during drying 6 hr. in air was only 5.2 per cent. The gain in weight by oxidation was therefore 1.9 per cent. The calorific value before oxidation when calculated to dry coal was 12,940 B.t.u., and the calorific value per pound of the dry, oxidized coal was 12,670 B.t.u., a loss of calorific value, when expressed in this way, of 270 B.t.u.

With a second coal, C-7, the moisture was 16.15 per cent, but the net loss of weight in drying was only 14.5 per cent, a difference of 1.65 per cent. The calorific values on the original weight showed an oxidation drop of 30 B.t.u., while expressed on the dry weight, before and after oxidation, the drop was 270 B.t.u. This treatment with these two coals therefore involved a notable loss of calorific value.

Measurement of Coal Oxidizability

Three methods have been employed in this laboratory to compare the oxidizability of different coals. In the first method calorimeter determinations were made on powdered coal before and after 6-hr. exposure to dry air at 106° C. The results were computed to the dry pure-coal basis.[5] The reduction in calorific value, in 11 coals tested, ranged from 60 to 520 B.t.u. per pound, or from 0.4 to 3.9 per cent reduction. The oxidized coal values are based on the weight of oxidized coal.

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[4] S. W. Parr and R. T. Milner: The Oxidation of Coal at Storage Temperatures. *Fuel* (1926) 301.

[5] E. Stansfield and J. W. Sutherland: Determination of Mineral Matter in Coal and Fractionation Studies of Coal. *Trans.* A.I.M.E. (1930) **88,** Coal Div. 623.

In the second method determinations of ulmins soluble in caustic potash were made on coals before and after exposure to air at 106° C. The percentage of soluble ulmins in one coal tested increased from 6.9 to 8.4 after 6 hr. oxidation, and to 46.6 after 600 hours.[6]

The third method consisted in the actual metering of the oxygen absorbed in coal. It was planned in an effort to make a simple, routine test for determining the rate of absorption of oxygen by coal exposed to air at ordinary temperatures.

The methods used by other workers[7] involved special apparatus and appeared to involve difficulties with sizing of the coal for test, prevention of oxidation in the preparation of the sample, etc. It was decided that the sample should not be fractionated by sieving, but that coarsely crushed coal should be taken and ground for a long period in a ball mill and the oxygen absorption determined during the grinding process. This reduced the risk of oxidation in preparation and insured an extremely finely divided coal and intimate contact with air during the test.

The apparatus consists of a gasometer filled with oxygen, humidity regulator, a Boys gas meter (containing oil instead of water), and a metal box in which the coal is ground in an Abbe porcelain ball mill. The gas meter and the box are joined by copper tubes soldered to the meter and box respectively, the two copper tubes being connected by an oil seal. The metal box has a removable cover which fits into an oil seal, the driving mechanism for the rollers on which the ball mill revolves is also oil-sealed. The lid of the ball mill has attached to it, inside the mill, a tube containing a porous, alundum thimble filled with soda lime, which serves to let gases enter or leave the mill without escape of coal dust, and to absorb any carbon dioxide given off by the coal. The mill chamber (metal box) contains more soda lime, also calcium nitrate to maintain the humidity at 45 per cent. The meter, metal box and humidity control bottles are placed inside a large box, with glass window, maintained at 30° C.

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[6] E. Stansfield and K. C. Gilbart: Determination of the Alkali-soluble Ulmins in Coal. *Trans.* A.I.M.E. (1932) **101**, 165.

[7] As, for example, the following:

H. C. Porter and O. C. Ralston: A Study of the Oxidation of Coal. U. S. Bur. Mines *Tech. Paper* 65 (1914).

M. C. Stopes and R. V. Wheeler: The Spontaneous Combustion of Coal in Relation to its Composition and Structure. *Fuel* (1923) 122.

G. Coles and J. I. Graham: The Influence of Preheating Oxidised and Partially Oxidised Coal upon its Rate of Absorption of Oxygen. *Fuel* (1924) 384.

W. Francis and R. V. Wheeler: The Oxidation of Banded Bituminous Coal at Low Temperatures. *Jnl.* Chem. Soc. (1925) **127**, 112.

S. W. Parr and R. T. Milner: Oxidation of Coal at Storage Temperatures. *Fuel* (1926) **5**, 298.

S. W. Parr: Deterioration and Spontaneous Combustion of Coal in Storage. *Fuel* (1926) **5**, 301.

D. J. W. Kreulen: The Change in Weight of Coal during Prolonged Oxidation at Low Temperatures. *Fuel* (1926) **5**, 345.

W. Francis and H. H. Morris: Relationship between Oxidisability and Composition of Coal. U. S. Bur. Mines *Bull.* 340 (1931).

A 200-gram sample of fresh coal, dried to about 45 per cent humidity in absence of air, is placed in the ball mill with a charge of steel balls, and the mill then put in the metal box and its cover replaced. The ball mill and box are filled with air, not oxygen. The wooden box is then closed and left until the temperature of the system is uniform at 30° C. The meter is then read and the ball mill rotated at about 18 r.p.m.

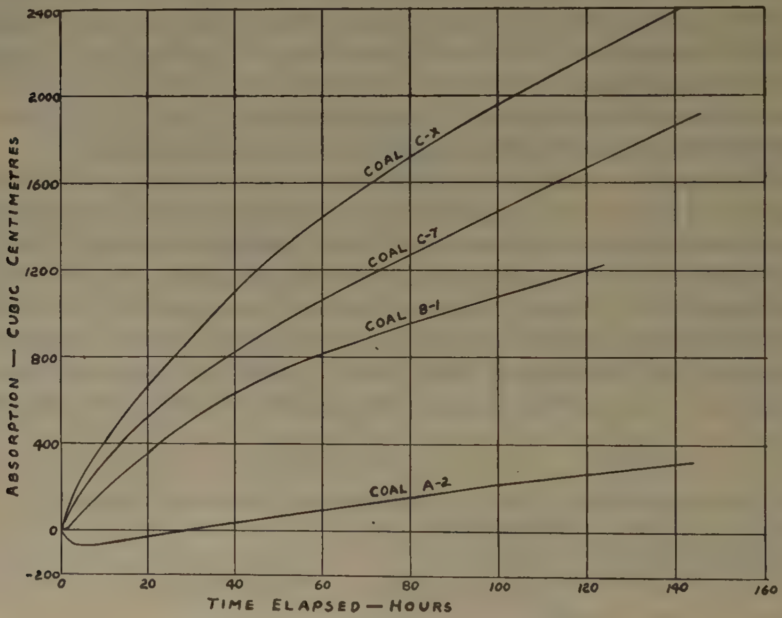

FIG. 1.—OXIDATION OF COAL IN AIR AT 30° C.
Oxygen absorption as cubic centimeters, at 30° C. and 700 mm., per 100 grams pure dry coal.

Oxygen is absorbed from the air in the mill but more oxygen enters from the gasometer, as shown by the meter, so that the oxygen content of the air remains fairly steady. A pressure of about 0.05 in. water is maintained in the oxygen gasometer but a three-way tap between the meter and the gasometer is opened before every meter reading to allow the pressure to drop to atmospheric. The meter readings are corrected for changes in barometric pressure by means of a table of corrections drawn up during a blank run. Readings are taken every ½ hr. at the start of the run and subsequently every 2 hr. during the daytime:

The course of oxidation of two coals, B-1 and C-7, is shown in Fig. 1. The rate of oxidation falls off rapidly for about 60 hr., but for the remainder of the test remains nearly uniform. The total absorption in the two coals during 120 hr. was equivalent to 1.47 grams and 1.98 grams respectively per 100 grams of dry, pure coal.

The National Research Council of Canada has given financial assistance in this work.

DISCUSSION

(*A. C. Fieldner presiding*)

W. A. SELVIG,* Pittsburgh, Pa. (written discussion).—The data given in this paper regarding (1) oxidation during storage of coal samples, (2) oxidation during air-drying, and (3) oxidation during the moisture determination, are of particular interest to the coal analyst in that they give specific information as to the errors that may occur in the methods now commonly used in this country.

The Joint Committee on Coal Analysis, of the American Society for Testing Materials and the American Chemical Society, in its preliminary report[8] recognized the instability of coal and the difficulties encountered in the determination of moisture. This committee recommended the use of an inert atmosphere for moisture determination in subbituminous coal and lignite. The committee also gave different methods for moisture; namely: (1) approximation methods and (2) methods of greatest accuracy. It was believed, however, that coals are so sensitive to humidity changes of the air that it seemed needless to strive in ordinary work for a high degree of refinement in the determination of moisture. Influenced by this viewpoint, the committee in its final report standardized on methods commendable for their simplicity, although open to some criticism from the standpoint of extreme accuracy.

Methods of accuracy are no doubt justified in scientific studies of coal, but it is debatable whether they are justified in the analysis of coal for commercial use.

MEMBER.—Is not coal C-7 high in fusain? In England this material is regarded as being highly dangerous from the standpoint of dust explosions because it absorbs oxygen rapidly.

E. STANSFIELD.—This coal, which is a subbituminous B coal from the Drumheller Valley, does not contain an unusually high percentage of fusain.

G. THIESSEN,† Urbana, Ill.—In connection with the grinding of samples, agglutination values are much more susceptible to oxidation than are moisture values, both during and after grinding. Certain border-line coals dropped 60 per cent in coking values after standing one day, then assumed constancy. This was with 100-mesh coal. I have used both CO_2 and natural gas for moisture determinations. CO_2 gave good results where air-drying failed. In duplicate tests between the standard method using air and in a closed system adapted for absorption of liberated moisture using air, loss by weight checked the standard method but absorbed moisture plus the dried coal was greater than 100 per cent, showing that oxygen had been absorbed. When CO_2 was used moisture absorbed plus dried coal totaled approximately 100 per cent.

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* Chemist, Pittsburgh Experiment Station, U. S. Bureau of Mines.

[8] *Ind. & Eng. Chem.* (1913) **5**, 517.

† Associate Chemist, Fuels Division, Illinois State Geological Survey.

Member.—Are the last curves diagnostic of different coals?

E. Stansfield.—They are. Not all of the methane came out after finer grinding and 48 hr. of evacuation. The oxidation affects the solubility in potash.

Member.—Was there evolution of heat in the ball mill?

E. Stansfield.—It was not detectable.

E. Stansfield (written discussion).—Dr. Selvig is of course correct in doubting the justification of exact methods in the analysis of coal for commercial use, where such exact methods could not readily be employed in commercial laboratories. On the other hand, as the A.S.T.M. definitely specifies the use of air in both the preliminary and final drying of coal, it might with reason be claimed that the more exact methods, which avoid oxidation, give incorrect results. This is an unfortunate state of affairs which we would like to see remedied.

Agglomerating and Agglutinating Tests for Classifying Weakly Caking Coals*

By R. E. Gilmore,† G. P. Connell‡ and J. H. H. Nicolls,‡ Ottawa, Ont.

(New York Meeting, February, 1934)

The purpose of this paper is to suggest a procedure for indicating the dividing line between noncaking coals and those that have weakly caking properties. A laboratory agglomerating test as an aid in classifying weakly caking coals is outlined and its merits in comparison with the standard agglutinating method advanced by the U. S. Bureau of Mines are discussed. No attempt, however, is made to discuss in detail either the different methods for determining "caking" and "agglutinating" indices or the various systems of classification.

Two methods, differing somewhat in procedure, have been employed and recognized as suitable for determining the caking or agglutinating values of coals; namely, those in which an inert material such as sand, electrode carbon, etc., has been mixed with the coal before being heated, and those in which the coal is carbonized alone. In all such methods, the coal with or without the inert material is in the finely divided state and the index or value sought is derived from the examination of the residue after heating in a closed crucible or other container. The procedure outlined by Selvig, Beattie and Clelland[1] is the most recent representative of the method using sand as inert material, while the agglomerating test described below is representative of the method in which the coal is carbonized alone.

The standard volatile-matter determination affords a means for observing the coking properties of a coal, and this has been described by Rose[2] as the simplest of all coking tests. According to him, "the resulting coke button will indicate whether the coal is noncoking, feebly coking or strongly coking," despite the fact that "almost every condition in this test is at variance with large-scale carbonizing practice." Lessing,[3] the originator of the carbonization assay test on the small laboratory scale, may be quoted as of the opinion that "the type of coal can be readily ascertained by well-known laboratory tests," and that "the appearance of the coke will give ready means for classifying each coal." In this connection, the following by Crussard[4] is of interest here:

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* Published by permission of Department of Mines, Canada.

† Superintendent, Fuel Research Laboratories and "Member-at-large" of Sectional Committee for the Classification of Coal.

‡ Assistant Engineer, and Chemist in Charge, respectively, Solid Fuels Analyses Section, Fuel Research Laboratories, Department of Mines.

[1] W. A. Selvig, B. B. Beattie and J. B. Clelland: Agglutinating Value Test for Coal. *Proc.* Amer. Soc. Test. Mat. (1933).

[2] H. J. Rose: Selection of Coals for the Manufacture of Coke. *Trans.* A.I M.E. (1926) **74,** 600.

[3] R. Lessing: The Carbonization of Coal (one of a series of lectures on coal and its utilization at the University of Sheffield). *Colliery Guardian* (1922).

> Among methods of observing the agglutination, the simplest is still the examination of the coke button from the rapid carbonization in a laboratory crucible. Whether the coke is swollen or not does not matter; one simply ascertains if it is in the form of powder, or of separate small aggregates or particles, or of a homogeneous, well-fused mass.

RELATIVE VALUE OF AGGLUTINATING INDICES FOR THE "USE" AND "SCIENTIFIC" CLASSIFICATION OF COALS

Of special interest is the classification scheme now being advanced[5,6] by the Sectional Committee on the Classification of Coal. A feature of this scheme is the provision of "groups" at the upper and lower boundaries of the bituminous "class," where agglutination is to be the deciding factor as to whether or not coals are to be included or excluded from the class. Such a provision is commendable, providing a definite understanding satisfactory to all concerned can be arrived at as to what coals are definitely nonagglutinating. Crussard,[7] whose findings as to the constituents of coal responsible for agglutination are of considerable interest, remarks that "agglutinating power is not the same thing as suitability for coking" and by coking he means the manufacture of coke commercially. The results[8] of comparative caking index tests by the Gray method and agglutinating value determinations by the Marshall-Bird method on a series of coals and blends, which were coked in the 2-ton byproduct oven at the Fuel Research Laboratories in Ottawa, failed to show any definite relation with the physical properties of the coke produced, and the relation of the indices obtained by the two methods was general only. Selvig[9] and his coworkers at the U. S. Bureau of Mines, after commenting that "it is probably asking too much to expect a laboratory agglutinating value test to correlate closely with the strength of commercial coke," quotes authorities to the effect that the test has value in connection with the detection of coking properties of coals during storage and affords information as to the tendency of coals to cake when used on ordinary steam-boiler grates and in domestic stokers.

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[4] M. L. Crussard: Agglutination of Coals and Its Relationship to the Action of Solvents. *Fuel* (1930) **9**, No. 4.

[5] W. A. Selvig, W. H. Ode and A. C. Fieldner: Classifications of Coals in the United States According to Fixed Carbon and B.t.u. See page 188, this volume.

[6] Report of Sectional Committee on Classification of Coals, 1934. *Preprint* Amer. Soc. Test. Mat.

[7] Reference of footnote 4.

[8] R. E. Gilmore: Relation of Caking Indices and Agglutinating Values of Coals to Their Laboratory and Plant Scale Coking Properties. Investigations of Fuels and Fuel Testing, 1929.

[9] Reference of footnote 1.

This "Bureau of Mines" method, which is now proposed as a tentative standard for adoption by the American Society for Testing Materials, does, as its authors claim, serve to show "significant differences in the agglutinating properties of various coals." It will be of value in selecting coals for different uses, provided that the agglutinating values or indices are correlated with the peculiar characteristics of the coals best suited for these uses. Its use, however, for the so-called "scientific" classification of coal appears to be somewhat limited. The fact that it was found difficult to obtain homogeneous buttons below the 14:1 ratio of sand to coal, coupled with the unevenness of the agglutinating curves of certain coals with ratios varying from this limit down to 8:1, leads one to the conclusion that the method is not as applicable to the weakly agglutinating coals as it is to those coals that contain sufficient agglutinating material to produce buttons that will not pulverize under a minimum weight (say 500 grams) at a ratio of sand to coal of 15:1. It would appear therefore that for coal-classification purposes this method would serve for ascertaining what coals are definitely caking rather than for indicating what coals are nonagglutinating, in the sense that they contain no agglutinating or agglomerating material whatever; that is, its use is limited for distinguishing between positive and negative caking.

AN "AGGLOMERATING" TEST FOR WEAKLY AGGLUTINATING COALS

The Fuel Research Laboratories at Ottawa have found that, by examination of the "coke buttons" obtained in the volatile-matter determination, it is possible to obtain valuable information as to the coking properties of coals. The practice followed is to grade the residue, after removal from the platinum crucible, in comparison with standard buttons from coals of which the characteristics are known. Five gradings at least are employed, which are recorded and reported in terms of general coking properties as follows: (1) noncoking (i. e., nonagglomerating), (2) agglomerating, (3) poor coking, (4) fair coking, (5) good coking.

The buttons from coals corresponding to these grades are shown in Fig. 1, each button being from one gram of coal. The "good" and "fair" coking gradings, it has been found, indicate individual coals or blends of coals that will produce a coke having physical properties ranging from fair to good, when coked in a commercial byproduct oven, while the "poor" coking grading indicates coals that will produce poor, very poor or no commercial coke. These gradings are therefore indications only of general commercial coking properties, and it has been observed that the grayish cokelike appearance of the buttons becomes more apparent as swelling increases, and that for the same grading the swelling of the coke button may vary considerably. Hence, the evaluation of a coal or coal blend by means of its "swelling index" determined on the coke button when carbonized at 600° C. as practiced[10] in the Carbonization Section of our Fuel Research Laboratories is important and necessary as a supplement to the coke-button gradings described above, especially for predicting the physical properties of the commercial coke product. The term "agglomerate" indicates coals that do not produce the typical grayish coke button, but which are considered to have weakly caking or agglutinating properties. The "noncoking (i.e., nonagglomerating)" grading is used only for coals that produce a noncoherent residue that can be poured out of the crucible as a powder or as flakes or lumps, that will pulverize easily with thumb and finger pressure.

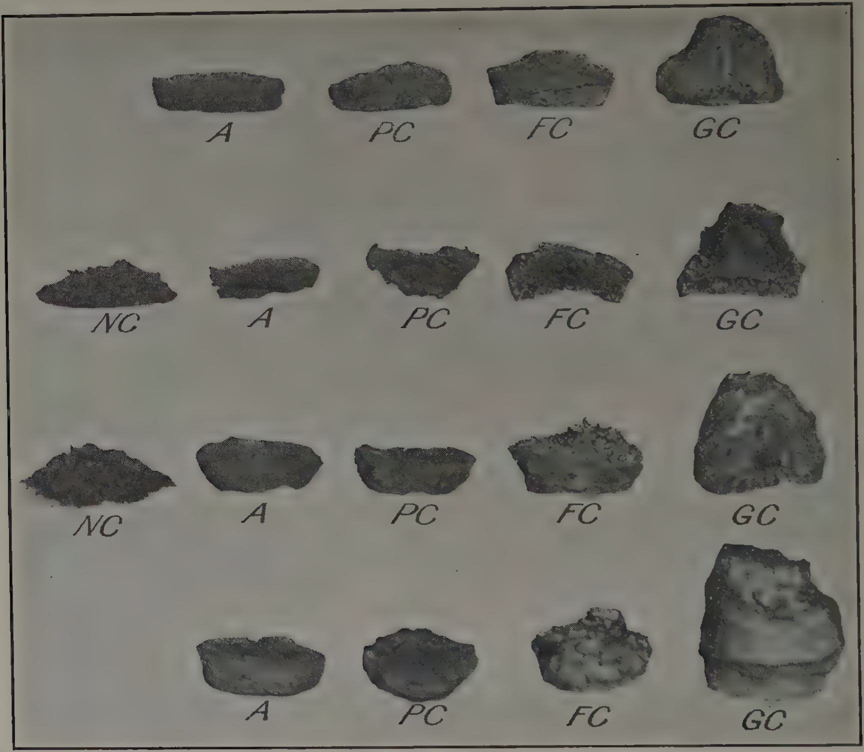

FIG. 1.—STANDARD "BUTTONS" FROM VOLATILE-MATTER DETERMINATIONS FOR JUDGING COMPARATIVE GENERAL COKING PROPERTIES OF COALS.

NC, noncoking (i.e., nonagglomerating).
A, agglomerating.
PC, poor coking.
FC, fair coking.
GC, good coking.

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[10] E. Swartzman, E. J. Burrough and R. A. Strong: A Laboratory Test on Coals for Predicting the Physical Properties of the Resultant By-Product Coke. Mines Branch No. 737-2, Canada (Investigations of Fuels and Fuel Testing, 1932).

Because of the nonuniformity of such a thumb and finger pressure test, especially to the inexperienced, a supplementary weight test has lately been employed for the "agglomerate" grading. The buttons that, after careful removal from the crucible, will pulverize under an ordinary 500-gram weight are judged and recorded as noncoking. In such a test the method of applying the pressure is to suspend the weight from the hands over the button, which is placed on the edge of the laboratory bench, and then, with the eye on the level of the top of the button to center the weight and to release so that it will not tip. Buttons (from weakly caking coals) that do not pulverize under the 500-gram weight, and that indicate no appreciable swelling are graded as "agglomerate."

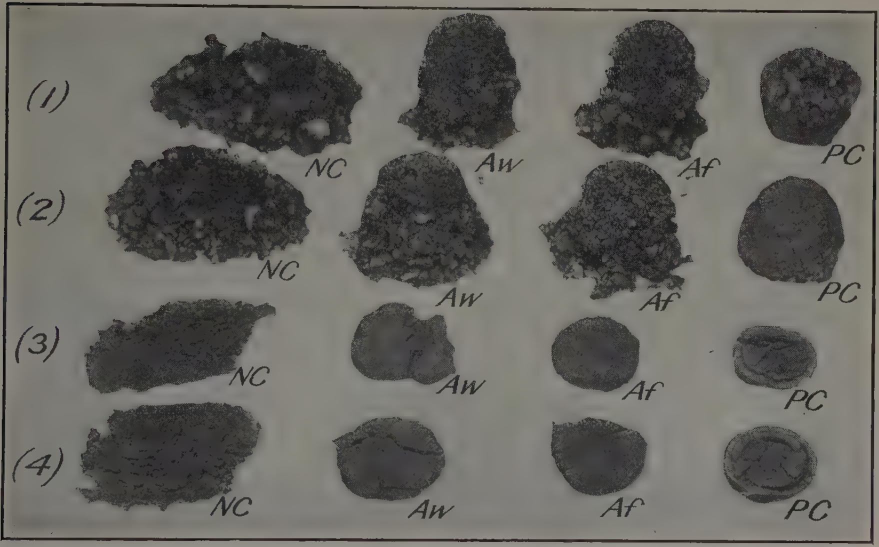

FIG. 2.—STANDARD "BUTTONS" TO ILLUSTRATE AGGLOMERATING PROPERTIES OF COALS.

Rows 1 and 2: Residues when 6 grams are heated in platinum crucible.

Rows 3 and 4: Buttons from 1 gram of the same coals in volatile-matter determination.

NC, noncoking (nonagglomerating).
Aw, weak agglomerating.
Af, firm agglomerating.
PC, poor coking.

The poor coking buttons, as illustrated in Fig. 1, show slight swelling and not such an opaque, black appearance as the agglomerate. The agglomerate buttons will in many cases support a 2000-gram weight or more without pulverizing, but under this pressure they mostly crack into smaller pieces. Such is the case with some of the poor coking buttons; hence, as a rule, all buttons with gradings better than "agglomerate" may crack but will not be pulverized under a minimum pressure of 2 kilograms.

In order to demonstrate the comparative appearance of buttons made from 1 gram of pulverized (ball mill) coal as used in the volatile-matter determination, and of the residue when approximately 6 grams of the crushed, unpulverized coal known as "coffee-mill size" are used, four coals ranging in coking properties from noncoking to poor coking having been selected, and the results shown in Fig. 2. Two of the four series shown pertain to the agglomerate grading and illustrate the difference between a weak (*Aw*) and the firm (*Af*) agglomerate buttons. The difference between weakly agglomerating and noncoking properties, on the left, and between the firmly agglomerating and poor coking properties, on the right, is also illustrated. Another purpose of Fig. 2 is to indicate how the agglomerating test may apply to the coffee-mill size of coal, as well as to the ball-mill material as generally used for proximate analysis. Such a test on the coffee-mill size is considered to be of value as confirmatory to that on the ball-mill size. Furthermore, coal in the pulverized state is more liable to oxidation and consequent deterioration in respect to general coking properties than in the coffee-mill size. For this reason the practice of keeping reserve samples of coals in the coffee mill size or larger should be followed and, in case of doubt, especially when freshly mined coal is not available, the agglomerating test may be repeated on this size or on freshly pulverized coal from the reserve sample.

RELATION BETWEEN AGGLOMERATE GRADING, CAKING INDEX AND AGGLUTINATING VALUE

Table 1 comprises the results of comparative tests on 14 standard coal samples showing agglomerate, poor, and fair coking properties, according to their coke-button gradings. Coals A, B, 1, 2 and 3 are high-rank coals, which would be classified on the dry basis according to the tentative classification mentioned above, while Nos. 4 to 10 inclusive, together with Y and Z, are lower rank coals and would be classified on the moist basis according to their calorific value in terms of B.t.u. per pound. The first four of the higher rank coals and the last six of the lower rank coals are considered border-line coals on the upper and lower boundaries respectively of the bituminous group. The analyses shown are on reserve coffee-mill samples rather than on freshly mined coal, so that these samples are to be considered only as standards for comparative tests and are not for the classification of the mines or areas in which they originated. The coke buttons from coals 1 to 10 inclusive are illustrated in Fig. 3 according to their respective gradings given in Table 1.

DISCUSSION OF RESULTS

The agglutinating determinations reported in Table 1 were conducted at the Pittsburgh Experiment Station of the U. S. Bureau of Mines, while the caking indices by the Gray method and the coke-button gradings were made at Ottawa. The significance of the values and gradings opposite each coal is explained in the footnotes and the relation of these gradings to the respective caking indices and agglutinating values are self-explanatory.

The agglutinating values in terms of kilogram pressures shown in Table 1, it may be emphasized, are on the 15 to 1 sand-coal ratio, and the figure of 0.5 kg. (500 grams) has been mentioned as the limit when using this ratio between the agglutinating bituminous coals and non-agglutinating lower and higher rank coals. Applying this limit to the coals listed in Table 1, we find that only four coals, 3, 4, 5 and 6, would be agglutinating and therefore be classed as bituminous. This would mean that weakly caking coals showing caking indices from slightly less than 1 up to 5 at least, and likely as high as 15 by the Gray method, would be ruled as nonagglutinating and not allowed as bituminous. This is evidently open to criticism and hence it would appear that either values lower than the 0.5 kg. at the 15 to 1 ratio, or lower sand-coal ratios, should be considered for adoption for general scientific classification purposes.

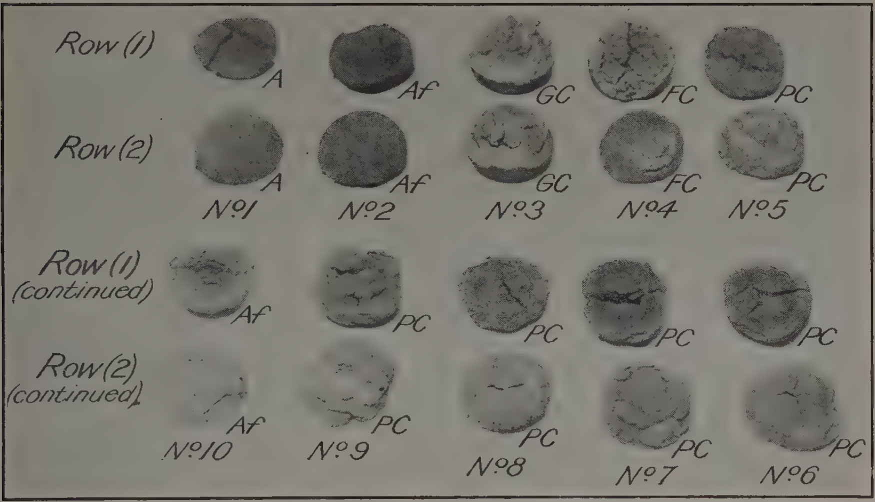

FIG. 3.—BUTTONS FROM THE VOLATILE-MATTER DETERMINATIONS CORRESPONDING WITH THE GRADINGS REPORTED IN TABLE 1.

Row 1: Buttons are right side up.
Row 2: Buttons are upside down.
Designations *A*, *Af*, *PC*, *FC* and *GC* are the same as in Figs. 1 and 2.

Mr. Selvig, in forwarding and commenting on the agglutinating value results on 10 of the standard coals reported in Table 1, mentioned that according to his observations as to the appearance of the residue coke buttons obtained in the volatile-matter determination, "coals 3, 4, 5, and 6, having the highest agglutinating values, all showed good fusion with pronounced cell structure"; that "the residue from coals 1 and 2

TABLE 1.—*Relation between Agglutinating Values, Caking Indices and Coke-button Gradings on Standard Coal Samples*

No.	Origin	Analysis of Coal (Mineral-matter-free),[a]		Agglutinating Value by U. S. Bureau of Mines Method[b]	Caking Index by Gray Method[c]	Coke-button[d] Grading to Indicate Coking Properties
		Fixed Carbon "Dry," Per Cent	B.t.u. per Lb. "Moist" (and Dry)			
A	Virginia	84.8				Noncoking
B	Wales	87.0		Crushed by 0.2 kg. (supported 0.1 kg.)	2	Agglomerate
1	Wales	85.6		Crushed by 0.2 kg. (supported 0.1 kg.)	5	Agglomerate
2	Alberta	83.8		Crushed by wt. of 0.1 kg.	1	Firm agglomerate
3	Alberta	82.2		Crushing strength 3.3 kg.	36	Good coking
4	Alberta	82.4		Crushing strength 2.1 kg.	47	Fair coking
5	Illinois	56.3	12,500 (14,500)	Crushing strength 1.9 kg.	20	Poor coking
6	Nova Scotia	57.0	12,510 (14,500)	Crushing strength 0.8 kg.	18	Poor coking
7	Nova Scotia	56.6	12,200 (13,800)	Crushed by 0.2 kg. (supported 0.1 kg.)	Less than 1	Poor coking
8	British Columbia	56.3	13,700 (14,800)	Crushed by wt. of 0.1 kg.	Less than 1	Poor coking
9	British Columbia	55.6	12,900 (14,300)	Crushed by wt. of 0.1 kg.	Less than 1	Poor coking
10	Alberta	58.2	13,700 (14,800)	Crushed by wt. of 0.1 kg.	Less than 1	Firm agglomerate
Y	Alberta	60.1	12,480 (13,870)			Noncoking
Z	Alberta	55.2	12,320 (13,690)			Noncoking

[a] "Dry" and "moist" are the oven-dry and moist (as received) bases respectively.

[b] The results in this column were from determinations made under the direction of W. A. Selvig, by permission of A. C. Fieldner. The agglutinating values reported are in terms of the "crushing strength" of the buttons, from the respective coals mixed with sand, in the 15 to 1 ratio, that is, 15 parts of sand to 1 part of coal, after being prepared and carbonized according to the U. S. Bureau of Mines method.

[c] These are the ratios of sand to coal to produce a carbonized button that will just support a 500-gram weight.

[d] The agglomerate and firm agglomerate buttons, as well as the poor, fair, and good coking buttons in this column all supported a 500-gram weight without pulverizing.

was sintered but showed no apparent cell structure"; and "it is questionable whether coals 7, 8, 9 and 10 gave residues in the volatile-matter determination which show any cell structure." These remarks illustrate the difference in viewpoint as to what agglutination really means. In the laboratories of the U. S. Bureau of Mines "agglutination" is seemingly closely associated with "cell structure" of the resultant button from the volatile-matter determination, whereas in (our) Canadian Fuel Research Laboratories the buttons graded as "agglomerate," "firm agglomerate" and "poor coking," which do not necessarily exhibit cell structure or appreciable swelling, are judged as weakly caking as opposed to noncaking. If the former viewpoint is to prevail, then only those coals producing buttons grading as good and fair coking together with some graded as poor coking will be classed as agglutinating bituminous coals, whereas according to the latter viewpoint those coals that form an agglomerate and all those graded as poor coking will be included. According to the literal and technical meanings of the terms, agglomeration of finely divided particles of coal implies agglutination, irrespective of the cell structure of the resultant button, and in the opinion of the writers, agglomeration signifies weak agglutination, and the agglomerate buttons typify weak agglutinating coals which should be classed within the bituminous group.

Let us sum up in tabular form (Table 2) the effect of these alternative viewpoints on the classification of the 14 coals listed in Table 1.

TABLE 2.—*Alternative Viewpoints on Classification*

Number of Coals with "Dry" Fixed Carbon and "Moist" B.t.u. Ranges	Classification in Bituminous Class Either in the Low-volatile Group or in the High-volatile C Group	
	Viewpoint 1 "Agglutinating" *Excludes* Weakly Caking Coals Graded as Agglomerating	Viewpoint 2 "Agglutinating" *Includes* Weakly Caking Coals Graded as Agglomerating[a]
Higher rank border-line coals A, B, 1 and 2 (Fixed carbon 83.8 to 87.0 per cent).	None	3 coals out of 4—Nos. B, 1 & 2—in low volatile group
Coals 3, 4, 5, and 6	All four coals	All 4 coals
Lower rank border-line coals 7, 8, 9, 10, Y and Z (B.t.u. 12,200 to 13,700).	None	4 coals out of 6—Nos. 7, 8, 9, & 10

[a] By "agglomerating" is meant buttons from the volatile-matter determination that will support a 500-gram weight, or have caking indices ranging from approximately 1 to 5.

The writers greatly favor viewpoint 2, not only because agglomerating and agglutinating both imply weakly caking, but because certain of the debatable coals (1, 2, 8 and 10) fall within the 86 per cent fixed carbon and 13,000 B.t.u. limits which automatically excludes them from the border-line group and allows them to be classified as bituminous.

STANDARD BUTTONS TO ILLUSTRATE COMPARATIVE COKING AND AGGLOMERATING PROPERTIES

The question may be raised as to whether it is possible to consistently arrive at a uniform grading of coke buttons resulting from the volatile-matter determination. It is quite possible for an inexperienced operator to wrongly judge and report an erroneous grading by simply looking at and fingering the buttons. As an aid to beginners, a series of standard buttons are retained with which new buttons may be compared. The set of buttons adopted as standard at the Fuel Research Laboratories includes more than the grading given above and comprises the following gradings; namely, noncoking, agglomerate, firm agglomerate, poor, poor to fair, fair, fair to good, and good. Such a set affords some leeway in comparative judging and, after at least two operators concur on the judgment, the detailed grading is recorded. In the laboratory report of analysis opposite the item "coking properties" the grading given is not usually as just detailed, but is according to the five grades mentioned above; namely, noncoking, agglomerating, poor, fair and good coking, and as illustrated in Fig. 1.

The "standard agglomerating test" to be recommended, it may be emphasized, is simply part of the laboratory procedure for indicating general coking properties of coals, by judging and grading the crucible residues, from the volatile-matter determination. An agglomerate button is arbitrarily defined as one that will support a 500-gram weight without pulverizing. Hence, buttons that fail to support this pressure represent the "noncoking" (i.e., nonagglomerating) grading and therefore only coals that in the volatile-matter determination fail to produce a button that will support 500 grams may be declared both nonagglomerating and nonagglutinating. It is also to be emphasized that in this test the coal is carbonized alone, without admixture of sand, and that such an agglomerating test as part of a system of coke-button gradings is to be considered as supplementary to, rather than replacing, other established caking-index and agglutinating-value tests in which sand as an inert material is mixed with the coal being carbonized.

In conclusion it is recommended that for the purpose of interpreting the terms agglutinating and nonagglutinating in the classification proposed for tentative adoption by the Sectional Committee on the Classification of Coal, where the coal is classified on the mineral-matter-free basis, the following meanings be implied:

All coals having less than 86 per cent fixed carbon on the dry basis and more than 13,000 B.t.u. per pound on the moist basis are to be considered as belonging to the bituminous group without reference to their agglutinating properties.

In the border groups (namely, the semianthracite group with dry fixed carbon limits of 86 to 92 per cent and the high-volatile C group with moist B.t.u. limits of 11,000 to 13,000), coals that will form carbonized sand-coal buttons not failing under 500 grams pressure at the 15 to 1 sand-cool ratio according to the procedure given in the U. S. Bureau of Mines method, and also those coals that in the volatile-matter determination will produce an "agglomerate" button that will support a 500-gram weight without pulverizing, are to be considered as "agglutinating" and therefore to be included within the bituminous class.

Acknowledgments

The writers are indebted to Mr. A. C. Fieldner, Chief of the Experiment Stations Division, U. S. Bureau of Mines, and Chairman of the (A.S.T.M.) Sectional Committee on the Classification of Coal, for instigating the preparation of this paper. To him and Mr. W. A. Selvig, and to their collaborators at the Pittsburgh (Bureau of Mines) Experiment Station, grateful acknowledgment is also due for their cooperation in making comparative agglutinating value tests on the standard coal samples reported in Table 1 and for their useful comments on the experimental results.

DISCUSSION

(H. J. Rose presiding)

W. A. Selvig,* Pittsburgh, Pa., commended the authors on the simplicity of the test, and the clarity with which they treated an involved subject. The Bureau of Mines method measures a greater degree of agglutination than does the Fuels Research Laboratory method. Recent tests on some nonagglutinating high-moisture mid-western coals showed them to be agglomerating by the F.R.L. method. There is need of further correlation between the two methods. Mr. Selvig added that Mr. Barkley, fuel engineer of the U.S. Bureau of Mines, is now correlating agglutinating values with caking properties of various coals when burned on grates.

R. L. Rowan,† Philadelphia, Pa., asked what studies had been made in correlating the agglutinating value test with the caking and coking action of various fuels in the fuel beds of grates and mechanical stokers.

In the eastern bituminous fields the extremely small sizes of coal swell and cake considerably more than the coarser fractions, such as nut and egg sizes when heat is applied to them in the fuel bed of these furnaces. Coals from various seams have different burning characteristics in stokers and grates, due to this coking or caking action. Coals from some seams are known as extremely light coking, others as extremely heavy coking or caking, and this action has a vital bearing on the burning characteristics of these coals.

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* Chemist, Pittsburgh Experiment Station, U. S. Bureau of Mines.

† Fuel Engineer, General Coal Co.

If the index of the agglutinating test can be correlated with this burning characteristic, it would be extremely valuable information to all concerned in the proper utilization and application of coal to various types of burning equipment.

A member stated that different fractions of a semianthracite behaved differently, the fusain being noncoking while the vitrain in some semianthracite coked. The lump coal might show no agglutinating power, although with some segregation of constituents taking place in sizing the finer sizes might show coking characteristics.

A. C. Fieldner* expressed the opinion that Mr. Gilmore's agglomerating test will find a place in the literature, whether or not it is used for border-line classification. The American Society for Testing Materials should adopt a standardized method of describing buttons, to lend uniformity to the interpretation of test results.

H. C. Porter,† Philadelphia, Pa., pointed out that the subcommittee on definitions of committee D-5 of the A.S.T.M. defined coke a year or two ago as a coherent cellular mass obtained as the residue of destructive distillation, with characteristic physical structure resulting from the solidification of a fused or semiliquid mass. Facetiously, Dr. Rose asked if bread would not meet these requirements, and Dr. Porter replied that bread could hardly be regarded as a "residue of destructive distillation."

R. E. Gilmore, in reply, commented further on the difference between the agglomerating and the agglutinating tests. The former he considered satisfactory for indicating weakly agglutinating or caking properties for the scientific classification of coals, whereas the latter was more suitable for demonstrating varying degrees of agglutination for purposes of use classification. In correlating the laboratory agglutinating test with burning characteristics in respect to caking, he believed the employing of an agglutinating value range for each individual use to be more practical than the selection of a definite minimum index such as 0.5 kg. at the 15 to 1 sand-coal ratio, for all uses, as undoubtedly the index representing the point where a coal begins to cake would vary considerably according to the type of burning equipment, size and kind of coal, draft conditions, and other factors. The gathering of data from coal producers and users as to the tendency and degree of caking in the actual use of their coal as marketed, he remarked, was a practice to be commended.

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* Chief Engineer, Experiment Stations Division, U. S. Bureau of Mines.

† Consulting Chemical Engineer.

Estimation of the Grindability of Coal*

By H. F. Yancey,† O. L. Furse,‡ and R. A. Blackburn,§ Seattle, Wash.

(New York Meeting, February, 1934)

For several years the Bureau of Mines at its Northwest Experiment Station, in cooperation with the Mines Department of the University of Washington, has been studying the breakage and degradation of coal. The first phase of the study was devoted to the determination of the friability of many coals of various ranks and comparison of various methods for estimating friability.[1]

A study of the relative ease or difficulty with which various coals are reduced to the fine size required for use as pulverized fuel followed the completion of this work. Baltzer and Hudson[2] list twelve research organizations in the United States, two in England, one in Germany and one in Canada that are reported to be engaged on this problem; showing the general appreciation of the need for a simple method of estimating the grindability of different coals. Hardgrove,[3] who has advanced a method for estimating grindability, points out that pulverizer tests of different coals are incomplete and unsatisfactory. He compares such tests to that of a boiler test made without knowledge of the calorific value of the coal. Frisch and Holder[4] also stress the same point. They state that

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* This paper presents the results of work done under a cooperative agreement between the Northwest Experiment Station, U. S. Bureau of Mines, and the University of Washington, Seattle, Wash. Published by permission of the Director, U. S. Bureau of Mines.

† Supervising Engineer, Northwest Experiment Station, U. S. Bureau of Mines.

‡ Research Fellow, University of Washington, 1932–33.

§ Research Fellow, University of Washington, 1933–34.

[1] H. F. Yancey, K. A. Johnson and W. A. Selvig: Friability, Slacking Characteristics, Low-Temperature Carbonization Assay, and Agglutinating Value of Washington and Other Coals. U. S. Bur. Mines *Tech. Paper* 512 (1932).

H. F. Yancey and R. E. Zane: Comparison of Methods for Determining the Friability of Coal. U. S. Bur. Mines *Rept. of Investigations* 3215 (1933).

[2] C. E. Baltzer and H. P. Hudson: A Method of Rating the Grindability or Pulverizability of Coal, Developed by the Fuel Research Laboratories (F.R.L.), Department of Mines, Canada. Dept. of Mines, Mines Branch, Canada *Rept.* 737-1 (1933) 1.

[3] R. M. Hardgrove: Grindability of Coal. *Trans.* Amer. Soc. Mech. Engrs., Fuels and Steam Power Div. (1932) **54**, 37; The Relation between Pulverizing Capacity, Power, and Grindability: Preprint for presentation at Spring Meeting of Amer. Soc. Mech. Engrs., Chicago, June 27, 1933.

[4] M. Frisch and G. C. Holder: Correlation of Grindability with Actual Pulverizer Performance. *Combustion* (1933) **4** and **5**, 29–34.

until recently it was not considered important to make fine distinctions between coals to be pulverized, but that the installation of equipment that failed to meet performance guarantees focused attention on the extreme differences in the grindability of apparently quite similar coals.

Previous Work

Descriptions of four laboratory methods of estimating the grindability of coal have been published; the Cross,[5] Gould,[6] Hardgrove[7] and F. R. L. (Fuel Research Laboratory, Canada).[8] In the Cross method, 200 grams of coal just passing 10 mesh is ground for 400 revolutions in the usual laboratory porcelain jar mill with a charge of eighty-nine ¾-in. steel balls.

Gould does not give the details of his method but he uses a scale of grindability ranging from 0 to 100, on which he indicates two grinding indexes for each coal, representing undercapacity and overcapacity mill performance. This investigator has made determinations on a large number of eastern and mid-continental coals, on the basis of which he divides bituminous coals into four groups according to these grinding characteristics.

The Hardgrove method requires a special grindability machine of the ring and ball type. A 50-gram sample of coal through No. 16 and on No. 30 sieve (U. S. series) is ground for 60 revolutions, using eight 1-in. steel balls.

The F.R.L. method requires the use of an initial feed sample of 500 grams stage-crushed by a special procedure just to pass 9-mesh Tyler or No. 10 U.S. series sieve. The sample is ground in the ordinary laboratory porcelain jar mill for 1000 revolutions with a charge of 3000 grams of 1 to ½-in. flint pebbles. The minus 100-mesh product is then screened out and the oversize, together with enough of the feed sample to replace the undersize, is ground a second time for 1000 revolutions. This process is repeated a third time, and the weight in grams then passing 100 mesh, together with the loss, is considered as the grindability index.

All these methods require grinding for a definite time. It is considered that the same amount of energy is expended on each coal. The differences between coals then are expressed by differences in the size composition of the resulting products. Both Cross and Hardgrove attempt to measure the new surface produced by screen-sizing the product. Baltzer and Hudson consider that the degree of reduction in size by the F.R.L. procedure is expressed by the percentage passing 100 mesh. Necessarily they must assume that the subsieve material through 100 mesh has the same mean size for every coal.

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[5] B. J. Cross: Hardness or Grindability Characteristics of Coal. Unpublished report of Subcommittee 7, Committee D-5, Amer. Soc. for Test. Mat.

[6] B. G. Gould: Factors in Coal Selection; Pulverizing Quality and Fuel Value. *Power* (1930) **72**, 886–889.

[7] Reference of footnote 3.

[8] Reference of footnote 2.

PRELIMINARY WORK BY BUREAU OF MINES

The preliminary work done by the Bureau of Mines followed closely that of previous investigators. Seven coals that had shown a wide range of friability were used in the grindability investigation. In general, three types of testing methods were employed. Two were laboratory ball-mill methods and one a drop-weight impact crushing method. The impact method seemed much the less promising and was soon discarded in favor of more intensive work with the two ball-mill procedures.

In early stages of the work the one-gallon, laboratory, porcelain-jar mill was employed, but its ease of breakage when used with steel balls and its nonuniform dimensions quickly forced the adoption of small steel mills of approximately the same size. Eight steel mills were constructed, and hundreds of grindability tests were made on different sizes of different coals under varying conditions of ball load, ball size and revolutions per minute. In this work somewhat irregular results were obtained, which could only be attributed to difference in the lubricative effects of coals.[9] It thus appeared that some means would have to be taken to eliminate the effect of the lubricative action on the smooth surfaces of the walls of the mill and the balls. Accordingly, four of the mills were equipped with three ribs 7/16 in. square; equally spaced and riveted longitudinally to the walls. Use of the ribbed mills enabled much more concordant results to be obtained. It seemed that a given number of revolutions then represented the same quantity of grinding work, irrespective of differences in the lubricative qualities of the different coals. This was not true with the smooth-sided mills. It was found also that the size of the feed used in the ribbed mill had much less effect upon the resulting grindability index than when the smooth mills were used.

The grindability index, by means of which the results of the different procedures were evaluated, was estimated by screen-sizing the mill product with every screen in the Tyler square-root-of-two series. To approximate surface, the reciprocal of the mean size between screens was multiplied by the weight per cent of that product and the summation was considered as the grindability index. The portion of the product that passed the 400-mesh sieve was assigned an assumed value of 25 microns. The mean size of this product and that between the different screen sizes was assumed to be equal to the square root of one-half the sum of the squares of the sides of the openings in the passing and retaining screens.

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[9] P. Rosin and E. Rammler: Ueber die Mahlbarkeit von Kohle. 25th *Berichtfolge* des Kohlenstaubausschusses des Reichskohlenrates. Berlin, 1931, Ver. deut. Ing. Verlag.

The same method was used for estimating the mean size of the material passing 400 mesh.

Careful study of the data accumulated in these experiments, with microscopic examination of the subsieve material that passed the finest screen (400 mesh), led to the unfortunate but inescapable conclusion that the results were unreliable, largely because of the assumed constant value of 25 microns for the subsieve sizes. When as little as 3 per cent of the mill product was finer than 400 mesh, this material represented 30 per cent of the total surface of the product. With larger percentages of subsieve material, as much as 90 per cent or more of the total surface is subsieve material. Since all subsieve material is not of the same size, estimation of grindability by any method based upon the calculation of the new surface produced is decidedly unreliable.

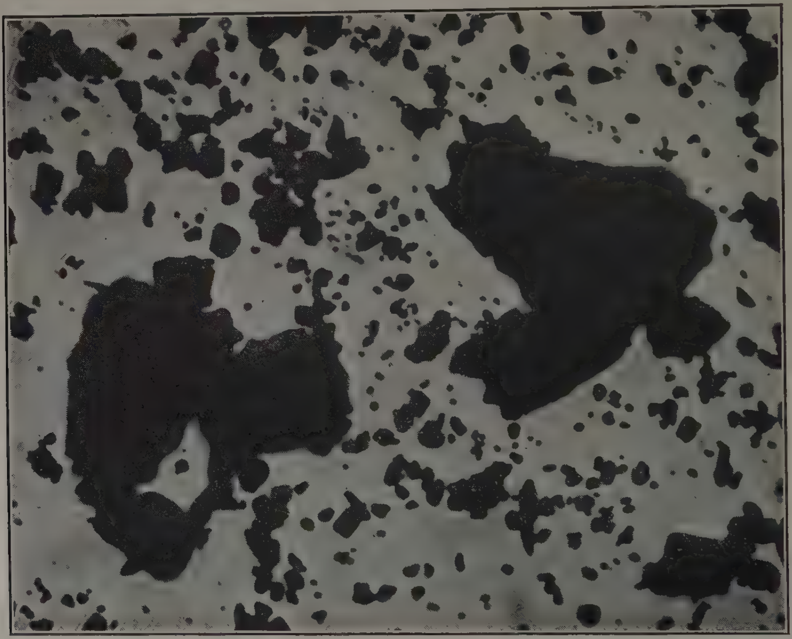

Fig. 1.—Photomicrograph of bituminous coal passing 400 mesh (36 microns). × 1000.

Microscopic counting of subsieve material is the best available method for estimating surface, but is not adaptable for use in connection with a simple laboratory method. Dye adsorption,[10] elutriation,[11] and air separation[12] are equally unsuitable. The rapid method of determining specific surface recently proposed by Wagner,[13] which utilizes the photoelectric cell for measuring turbidity, may prove suitable for this purpose.

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[10] R. A. Sherman, C. E. Irion and E. J. Rogers: A Study of Adsorption as a Method for the Determination of the Surface of Pulverized Coal. Preprint, Amer. Soc. Test. Mat., Chicago, Ill., June 26–30, 1933.

[11] J. Gross, S. R. Zimmerley and A. Probert: A Method for Sizing of Ore by Elutriation. U. S. Bur. Mines *Rept. of Investigations* 2951 (1929).

TABLE 1.—*Proximate Analyses of Coals Used in Grindability Investigation*

Coal				Analysis, Per Cent				
State	County	Mine	Bed	Basis[a]	Moisture	Volatile Matter	Fixed Carbon	Ash
Washington...	Pierce	Wilkeson	No. 7	1	0.9	25.6	61.4	12.1
				2		25.8	62.0	12.2
				3		29.4	70.6	
Washington...	Pierce	Wilkeson	No. 4	1	1.1	35.8	53.8	9.3
				2		36.2	54.4	9.4
				3		40.0	60.0	
British Columbia.	Vancouver Island	Ladysmith Wellington No. 5	Alexandria	1	2.0	39.1	50.8	8.1
				2		39.9	51.8	8.3
				3		43.5	56.5	
Washington...	King	Bayne	Carbon	1	2.4	32.0	48.3	17.3
				2		32.8	49.5	17.7
				3		39.9	60.1	
Washington...	Kittitas	Roslyn 3	Roslyn	1	2.4	36.8	45.4	15.4
				2		37.7	46.5	15.8
				3		44.8	55.2	
Utah.........	Carbon	Rolapp	Castlegate	1	2.3	41.1	49.9	6.7
				2		42.1	51.1	6.8
				3		45.2	54.8	
Washington...	Whatcom	Bellingham	Bellingham	1	5.4	38.7	42.2	13.7
				2		40.9	44.6	14.5
				3		47.8	52.2	
Pennsylvania..				1	2.5	3.2	84.5	9.8
				2		3.3	86.6	10.1
				3		3.7	96.3	

[a] 1, sample as tested; 2, dried at a temperature of 105° C.; 3, moisture-and-ash-free.

Some of the difficulties inherent even in the microscopic methods of estimating size and surface may be visualized by referring to Fig. 1, which shows the wide range of sizes and shapes in a coal passing 400 mesh. Similar photographs were taken of the through 400-mesh fraction in identical grinding tests of 14 other coals. Although this work is incomplete, it seems evident that coal passing 400 mesh has an average surface-mean size of about 10 microns instead of 25 microns, the assumed value here used. Moreover, the mean size of this subsieve material is not the same for different coals. This is further confirmation of the inaccuracy of using assumed values for subsieve material in surface calculations for estimating the grindability of coal.

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[12] P. S. Roller: Separation and Size Distribution of Microscopic Particles. U. S. Bur. Mines *Tech. Paper* 490 (1931).

[13] L. A. Wagner: A Rapid Method for the Determination of the Specific Surface of Portland Cement. *Proc.* Amer. Soc. Test. Mat. (1933) **33,** Pt. II.

The possibility that the accuracy of the surface method of estimating grindability might be greatly increased by shortening the grinding period was investigated and rejected because of two considerations: (1) Even when 3 per cent of the feed was minus 400 mesh it represented approximately 30 per cent of the total surface; (2) in the short grinding period only the softest or least resistant portion of the coal is affected.

Proposed Bureau of Mines Method

The considerations just discussed led to abandonment of all attempts to develop a method for estimating grindability based on the calculation of new surface produced. Since, according to the Rittinger equation, the energy required for crushing is proportional to the new surface produced, it was decided to attempt evaluation of grindability by crushing each coal to the same useful fineness and determining the energy required to give this end point.

Table 1 shows the source and proximate analysis of the eight coals of which grinding characteristics were studied. They include a low-volatile anthracite from Pennsylvania, five bituminous coals from Washington (ranging in rank from the top to the bottom of the bituminous scale and from very high to zero in agglutinating value), and two other bituminous coals, one from Utah and one from Vancouver Island, British Columbia. The coals cover a wide range in grinding characteristics.

Procedure

Mills.—Ribbed mills (Fig. 2) were used in this work. They are 8 in. long, 8⅛ in. in diameter (inside dimensions), ¼ in. thick, and contain three equally spaced 7⁄16-in. square lifters riveted to the inside of the shell. They were made from sections of ordinary 8-in. steel water pipe with a lid and cover of ¼-in. steel plate. The bottom was welded to the shell, and the top was fitted with a rubber gasket and crossbar. The ball charge is made up of 100 alloy-steel ball bearings, each 1 in. in diameter. The device for rotating the mills is the ordinary frame commonly used for porcelain-jar mills. It is necessary to lengthen slightly the brass straps that support the mills in the frame because of the somewhat greater outside diameter of the steel mills. A speed of 40 r.p.m. is used.

Preparation of Coal Sample.—A 500-gram sample of air-dried coal, crushed to pass a 10-mesh Tyler sieve, is used. The procedure of graded roll crushing used in the F.R.L. method is followed with the modification that the final setting of the rolls is made at 1⁄32 of an inch instead of having the rolls just touch each other. In this system the sample is reduced by a succession of crushings with the roll clearance set to ½, ¼, ⅛, 1/16, and 1/32 in. Finer than 10-mesh material is removed by screening after each crushing and only the oversize is crushed at the next closer roll setting. Although in the work described here the material passing 200 mesh was not removed from the samples, it is recommended that this procedure be modified to include such removal, as a precaution against overcrushing.

Method of Test.—A 500-gram sample of coal through 10 mesh obtained by riffling the lot previously crushed to pass 10 mesh is placed in the ribbed mill with one hundred 1-in. steel balls. The mill is closed and rotated at 40 r.p.m. in the jar-mill frame until approximately 10 per cent of the sample is fine enough to pass 200 mesh (Tyler). This number of revolutions, which is termed a "cycle," may be determined by experiment. At the end of the cycle the contents of the jar, including the balls, are poured on to a sieve coarse enough to retain the balls but not the coal. The jar and balls are brushed to remove the sample, which is then carefully hand-screened on a 10-in. sieve until not more than ½ gram passes the 200-mesh sieve in one minute of screening. Both the oversize and the undersize are weighed; and the loss, which usually ranges from 1 to 3 grams, is considered as through 200-mesh material. The oversize is returned to the mill for the second cycle of grinding, which consists of the same number of revolutions as that originally required to give about 10 per cent (50 grams) of finished product. The mill is then stopped and the finished product is screened out as before. This same procedure is continued until at least 80 per cent (400 grams) of the original sample is obtained as through 200-mesh material.

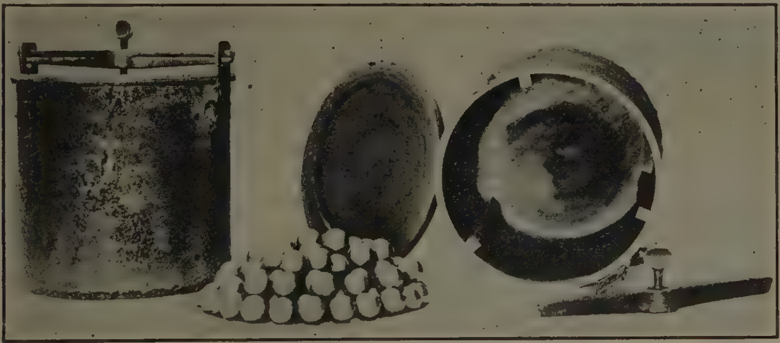

FIG. 2.—BALL MILLS USED FOR ESTIMATING GRINDABILITY OF COAL BY PROPOSED BUREAU OF MINES METHOD, SHOWING BALL CHARGE AND RIBS TO PREVENT SLIPPAGE OF BALLS.

Table 2 shows the data obtained on one coal with this method of grinding. The end point of the test is taken as the number of revolutions required to grind the coal so that 80 per cent passes a 200-mesh sieve. This number may be determined by interpolation of the data, or by plotting the cumulative percentage through 200 mesh against the number of revolutions.

The results may be expressed either as the number of revolutions required to grind the sample to a definite fineness (say 70 or 80 per cent through 200 mesh) or as the rate of crushing, in pounds per hour. Inasmuch as pulverizer capacity generally is stated in terms of units of weight and time—that is, in tons or pounds per hour of finished product of a certain fineness, say 70 or 80 per cent through 200 mesh—it is believed that grindability comparisons of different coals will prove most useful if expressed in the same units. For convenience in calculation, the rate in pounds per hour may be obtained by dividing the factor 2645 by the number of revolutions.

TABLE 2.—*Data Obtained in Grindability Test of Pennsylvania Anthracite by Proposed Bureau of Mines Method*

Total Number of Revolutions	Weight through 200 Mesh, Grams	Weight, Per Cent	Cumulative Weight, Per Cent	Ash,[a] Per Cent	Cumulative Ash,[a] Per Cent
250	53	10.6	10.6	13.2	13.2
500	51	10.2	20.8	11.7	12.5
750	51	10.2	30.0	11.0	12.4
1000	51	10.2	40.2	10.5	11.9
1250	50	10.0	50.2	10.1	11.5
1500	51	10.2	60.4	9.9	11.3
1750	47	9.4	69.8	9.5	11.0
2000	47	9.4	80.2	9.2	10.7
2250	45	9.0	89.2	8.9	10.5
2250	54[b]	10.8	100.0	8.3	10.3

[a] Moisture-free basis.

[b] Residue coarser than 200 mesh.

RESULTS OF INVESTIGATION

Table 3 shows the grindability of eight coals, expressed in pounds per hour of finished product, 70 and 80 per cent of which will pass a 200-mesh sieve, and by the number of revolutions required to give these finenesses. The variation in grindability ranges from 1.33 lb. per hour of pulverized fuel, 80 per cent of which will pass 200 mesh, for Pennsylvania anthracite, to 4.37 lb. per hour at the same fineness for the high-rank bituminous Wilkeson, bed No. 7, coking coal from Washington. Thus the anthracite offers over three times the resistance to pulverization to the same fineness as the Wilkeson 7 coal, or for a given expenditure of energy, 3.29 tons of Wilkeson coal can be pulverized to the same fineness in the same time as one ton of anthracite. Other grindability indexes and ratings previously advanced may not be so readily and usefully interpreted.

TABLE 3.—*Grindabilities of Eight Coals by Proposed Bureau of Mines Method*

Coal	Rate, Lb. per Hr. of Product		Revolutions Required for	
	70 Per Cent through 200 Mesh	80 Per Cent through 200 Mesh	70 Per Cent through 200 Mesh	80 Per Cent through 200 Mesh
Pennsylvania anthracite	1.52	1.33	1740	1990
Bellingham	1.70	1.56	1560	1700
Utah	2.04	1.76	1295	1500
Roslyn	2.18	1.86	1215	1425
Bayne	2.57	2.19	1030	1210
Wellington	2.91	2.47	910	1070
Wilkeson 4	3.15	2.67	840	990
Wilkeson 7	5.29	4.37	500	605

Fig. 3 shows, in graphic form, the complete results of the tests. It will be observed that the plotting of cumulative per cent passing 200 mesh against the number of revolutions of the mill gives a straight line up to 50 per cent or more through 200 mesh, depending upon the grinding char-

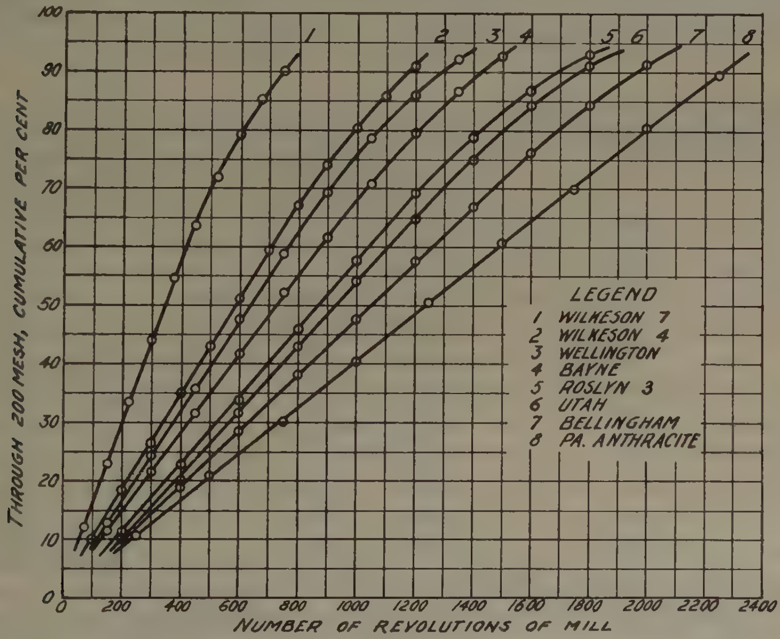

FIG. 3.—COMPARATIVE GRINDING RATES OF DIFFERENT COALS, DETERMINED ACCORDING TO PROPOSED METHOD OF ESTIMATING GRINDABILITY.

acteristics of the particular coal. This indicates a direct relationship between the work done as represented by the number of revolutions and the percentage passing 200 mesh. The decrease in slope of the curves as more of the coal is ground probably is due principally to the decreased weight of unfinished material in the mill and also the accumulation of material that is harder to grind.

DISCUSSION OF RESULTS

Reproducibility of Method.—Table 4 shows the results of check determinations on two of the coals made at different times. The operator who conducted the tests had no knowledge of the results he had obtained previously. The tests of Wilkeson 7 coal were made under three slightly different conditions. In two trials, test 1 and test 2, different feed samples from the same lot of coal were used. The testing conditions were identical in each case. In both, 75 revolutions constituted a cycle. A third trial, test 3, was made on the same feed sample as test 1, but at 100 revolutions per cycle instead of 75. This series on the Wilkeson coal therefore constitutes a rather severe test of the reproducibility of the method. As shown in the table, the number of revolutions required to give a product 80 per cent of which passed 200 mesh for the tests 1, 2, and 3 are 600, 605, and 620, respectively. The agreement is considered satisfactory.

TABLE 4.—*Check Determinations of Grindability by Proposed Method*

Cycle	Through 200 Mesh, Cumulative Per Cent				
	Wilkeson 7 Coal			Bellingham Coal	
	Test 1[a]	Test 2[b]	Test 3[c]	Test 1[d]	Test 2[d]
1	11.6	12.0	15.0	9.4	9.4
2	22.8	22.8	29.6	18.2	18.2
3	34.0	33.4	43.6	27.6	27.4
4	44.8	44.0	57.0	37.0	37.2
5	55.4	54.6	68.4	46.6	47.0
6	64.6	63.4	78.4	56.6	57.0
7	73.2	71.8	86.4	66.2	66.0
8	80.6	79.0	92.2	75.0	75.0
9	86.6	83.2		83.4	83.4
10	91.4	90.0		90.2	90.2
11				95.4	95.4
Revolutions to give 80 per cent through 200 mesh..............	600	605	620	1720	1720

[a] 75 revolutions per cycle; ash content of feed, 10.2 per cent.
[b] 75 revolutions per cycle; ash content of feed, 12.2 per cent.
[c] 100 revolutions per cycle; same feed sample as test 1.
[d] 200 revolutions per cycle; identical feed samples in both tests.

More concordant results are obtained on "hard" coals than on "soft" coals because the soft coals are more difficult to screen. This point is illustrated in the test of Bellingham coal, also shown in Table 4, where practically identical results were obtained in each of the successive cycles throughout the experiment. It is evident that the accuracy of the proposed method depends upon the assumption that the finished product removed from the mill at the end of each cycle for the same and for different coals has the same surface-mean diameter. This is probably not wholly true. For it to be even approximately true would require removal of the 200-mesh material just as soon as it was produced in the mill. This condition is extremely difficult, if not impossible, to fulfill in a laboratory procedure. Restriction of the length of the cycle to 5 per cent of finished product for soft coals before its removal by screening might improve somewhat the accuracy of the procedure. Nevertheless the writers believe that the errors in the proposed method are significantly less than those in previously proposed procedures.

Removal of Product.—The restriction of length of cycle of grinding to a relatively short period before removing the finished product prevents excessive grinding of finished material. The product of each cycle must tend to be of the same mean size. This short period of grinding and the successive removal of finished material from the system also serve another important function, allowing the oversize to be crushed rapidly at a fairly constant rate.

Fig. 4 illustrates the great difference in rates of grinding occasioned by removing the finished product at short intervals and by permitting it to remain in the mill. Duplicate samples of the same coal feed were used. In one test the material passing 200 mesh was removed by sieving after each cycle of 200 revolutions. In the other, the material was screened out and weighed but returned to the mill after each cycle. After 1400 revolutions, or 7 cycles, the finished product amounted to 78.4 per cent where it was rejected as compared with only 53.0 per cent for the same number of revolutions where it was retained in the mill.

This is the reason for the modern practice of closed-circuit grinding. It is also a reason why coal grindability ratings or indexes employing relatively long and constant periods of grinding cannot give true results. Cushioning action varies with the resistance of the particular coal to grinding and is dependent upon the rate of production of fine sizes. This rate is different with different coals. Particular attention and study have been given by Hardgrove[14] to the cushioning action of fine material and its effect on the relation of grindability to the capacity of pulverizers.

Ash Content of Products.—Table 5 shows the ash contents of the feed and the finished products passing a 200-mesh sieve obtained in each cycle.

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[14] Reference of footnote 3.

There is a general tendency for the ash contents of a given coal to increase as the process of grinding continues. The Utah coal and the Pennsylvania anthracite present exceptions in some of their products. In general, the data shown conform to the known belief that the bright low-ash bands of coal are more friable than the dull, higher-ash bands. The uncrushed oversize residue, except for the anthracite, is in all cases higher in ash than the feed. One of the chief advantages of the proposed method is that each coal, regardless of the proportions of "hard" and "soft" material present, is equally ground to the same end point, 70 or 80 per cent through 200 mesh, whichever figure is chosen.

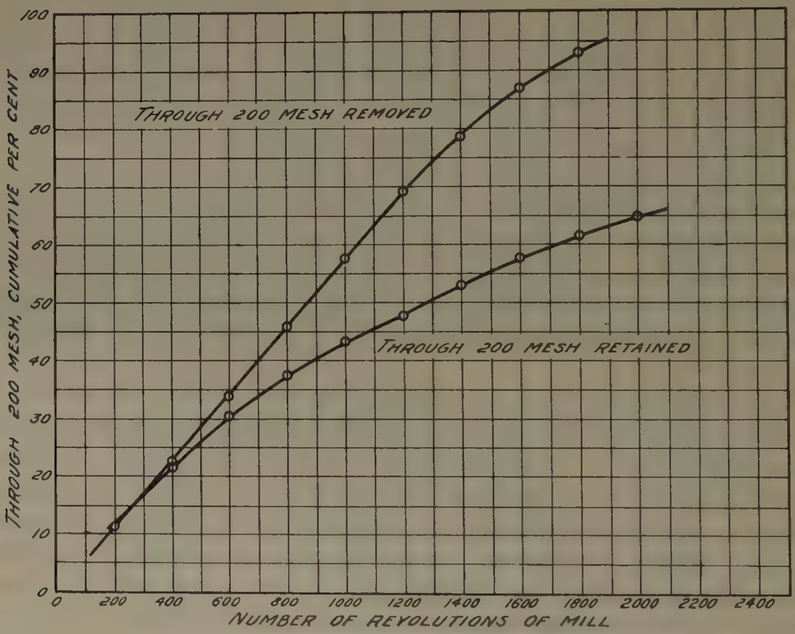

FIG. 4.—REDUCTION IN RATE OF GRINDING DUE TO RETENTION IN MILL OF MATERIAL PASSING 200 MESH.

Table 6 shows direct evidence of the variation in resistance to grinding offered by different specific gravity fractions of the Wilkeson 7 coal used in this investigation. A different sample than that used in the regular grindability studies was obtained from the mine and separated into five specific-gravity fractions by means of four float-and-sink baths of zinc chloride dissolved in water. This is the ordinary float-and-sink method used for estimating the washability of coal. The float-and-sink test was made on a 250-lb. head sample sized between 2 in. and 20 mesh. The five float-and-sink products were stage-crushed to pass a 10-mesh Tyler sieve by the procedure outlined.

Table 6 shows that the number of revolutions required to give a product 80 per cent of which will pass a 200-mesh sieve increases with increase in specific gravity of the particular fraction. The material in this particular coal, which is less than 1.30 sp. gr., contains 4 per cent of ash and requires only 520 revolutions to crush 80 per cent of it through 200 mesh. The material with a specific gravity greater than 1.70 contains 58 per cent ash and requires 1115 revolutions, or more than twice as many as the float 1.30 sp. gr. product. A composite sample of the float-and-sink products, made up in accordance with the percentage

TABLE 5.—*Ash Contents of Finished Products, Through 200 Mesh, Produced in Each Cycle of Proposed Method*

	PERCENTAGE, MOISTURE-FREE BASIS							
Cycle[a]	Wilkeson 7	Wilkeson 4	Wellington	Bayne	Roslyn 3	Utah	Bellingham	Pennsylvania Anthracite
Feed	12.2	9.4	8.3	17.7	15.8	6.8	14.5	10.1
1	10.6	9.1	8.8	16.0	15.8	8.4	15.0	13.2
2	10.1	8.5	8.0	15.4	14.7	7.2	14.0	11.7
3	10.1	8.5	7.8	15.5	14.7	6.7	13.8	11.0
4	10.3	8.4	7.7	15.6	14.7	6.4	13.8	10.5
5	10.6	8.7	7.7	15.9	15.0	6.3	13.9	10.1
6	11.3	8.7	7.7	16.3	15.5	6.3	14.1	9.9
7	11.7	8.8	8.0	17.0	16.2	6.3	14.3	9.5
8	12.7	8.9	8.3	17.9	17.0	6.7	14.6	9.2
9	14.0	9.6	8.8	19.4	18.2	7.1	15.0	8.9
10	15.6	9.8	10.2[b]	21.5	20.6[b]	9.4[b]	15.5	8.3[b]
11	21.3[b]	10.1		27.1[b]			16.7[b]	
12		10.7						
13		13.4[b]						

[a] The number of revolutions in a cycle for each coal is as follows: Wilkeson 7, 75; Wilkeson 4, 100; Wellington, Bayne, 150; Roslyn 3, Utah, Bellingham, 200; Pennsylvania anthracite, 250.

[b] Ash content of the oversize residue.

TABLE 6.—*Grindability of Specific-gravity Fractions of Wilkeson 7 Coal by Proposed Method*

Specific Gravity	Weight, Per Cent	Ash, Per Cent	Cumulative Weight, Per Cent	Grindability, Revolutions	Cumulative Grindability, Revolutions
Under 1.30	25.6	4.0	25.6	520	520
1.30 to 1.40	44.9	11.9	70.5	600	571
1.40 to 1.50	16.9	23.9	87.4	760	608
1.50 to 1.70	6.2	35.9	93.6	890	626
Over 1.70	6.4	58.0	100.0	1115	658
Composite sample	100.0	16.7		635	

composition shown, had a grindability value of 635 revolutions as compared with a grindability of 658 revolutions given by combining the individual grindabilities by calculation. This may be regarded as a drastic test of the accuracy of the method. The difference between these two values is only 23 revolutions.

Effect of Size Composition of Feed upon Grindability

It is obvious that there will be more or less variation in the size composition of different coals after crushing to pass 10 mesh, even though the procedure of stage-crushing by rolls previously described is strictly followed. Table 7 shows the screen analyses of the head samples of each of the eight coals recorded in this report. The principal differences in size composition between the coals are in the two coarser sizes, 10 to 14 mesh and 14 to 20 mesh. In order to determine the variation in grindability values that might be expected in various laboratories through differences in the application of the preliminary crushing method, a series of tests was made, in which the 10 to 14-mesh portion of the feed varied from 25 to 100 per cent, in steps of 25 per cent. Table 8 shows the results. The maximum variation is from 1370 revolutions for 25 per cent of 10 to 14-mesh size to 1550 revolutions for 100 per cent of 10 to 14-mesh size. This is a difference of 180 revolutions. Such wide variations of the feed size, even in different laboratories, are of course impossible. One of the reasons for variations due to differences in size composition is caused by the material in the feed which is finer than 200 mesh. This material appears as finished product at the end of the first cycle. An increase in the accuracy of the proposed method should result from the removal of the through 200-mesh material from the preliminary sample prior to the start of the grindability tests.

TABLE 7.—*Size Composition of Through 10-mesh Feed Samples Used in Grindability Tests by Proposed Method*

Size,[a] Mesh	Wilkeson 7	Wilkeson 4	Wellington	Roslyn 3	Bayne	Bellingham	Utah	Pennsylvania Anthracite
10 to 14	45.0	35.4	38.0	47.4	36.4	38.4	34.0	43.2
14 to 20	17.4	19.2	21.4	20.8	21.0	23.4	27.6	23.4
20 to 28	10.4	15.0	14.6	10.8	13.8	13.4	15.4	12.6
28 to 35	8.4	10.6	8.8	7.2	9.2	9.0	9.0	7.6
35 to 48	5.4	6.2	5.2	4.4	5.8	4.8	4.8	4.0
48 to 65	5.0	5.8	4.6	4.0	5.4	4.6	4.0	3.4
65 to 100	2.2	2.2	2.0	1.6	2.2	1.8	1.4	1.6
100 to 150	1.8	2.0	1.6	1.4	2.0	1.6	1.4	1.2
150 to 200	1.6	1.6	1.6	1.2	1.6	1.2	0.8	1.2
Through 200	2.8	2.0	2.2	1.2	2.6	1.8	1.6	1.8
	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0

[a] Tyler standard sieves.

TABLE 8.—*Effect of Difference in Size Composition of Feed on Grindability of Roslyn 3 Coal*

Size Composition of Sample	Number Revolutions to Give 80 Per Cent through 200 Mesh	Rate, Lb. per Hr.
10 to 14 m., 25 per cent Through 14 m., 75 per cent	1370	1.93
10 to 14 m., 50 per cent Through 14 m., 50 per cent	1440	1.84
10 to 14 m., 75 per cent Through 14 m., 25 per cent	1460	1.81
10 to 14 m., 100 per cent	1550	1.71

COMPARISON OF PROPOSED BUREAU OF MINES AND F.R.L. METHODS

The F.R.L. method[15] of rating the grindability of coal developed by the Fuel Research Laboratories, Department of Mines, Canada, has been correlated with the Cross and Hardgrove methods by the Canadian Laboratories. They found that there appears to be a fairly uniform relation between the indexes obtained by their method and the other two and concluded that any one may serve as a standard method for determining the grindability index of coal. Therefore, it was considered advisable to test the same coals examined in this investigation by the F.R.L. method.

Table 9 shows a comparison of the F.R.L. and the proposed Bureau of Mines methods, showing both the actual and relative values. The order of grindability is the same in both methods. The relative range of the F.R.L. method is from 23 to 100, compared with a range of 30 to 100 for the Bureau of Mines method; thus, a greater range is shown by the F.R.L. procedure.

Close inspection of the relative values given in Table 9 reveals significant differences between the two methods. For example, the grindabilities of Utah, Roslyn, and Bayne coals 60, 61 and 64 are not significantly different according to the F.R.L. method. The proposed method, however, gives values of 40, 43 and 50, showing significant differences between these three coals. In fact, the entire range covered by the seven bituminous coals is only 100 to 55, according to the F.R.L. method, compared with a range of 100 to 36 by the Bureau of Mines method. The difference in range between the two methods is thus 45 and 64 units, respectively. In other words, the proposed method separates the different coals much more decisively than does the F.R.L. procedure. Obviously, the reason for this effect is due to the cushioning action of the finished product produced in the early stages of the grinding operation according to the F.R.L. procedure. This action, previously discussed, and shown in Fig. 4, not only decreases the rate at which material finer than 100 mesh is produced by the F.R.L. method, but also causes the subsieve material to be ground to a greater extent with one coal than with another. Thus the units of the F.R.L. index are not of the same magnitude throughout the range from 0 to 500.

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[15] Reference of footnote 2.

TABLE 9.—*Comparison of F.R.L. and Proposed Bureau of Mines Methods for Estimating the Grindability of Coal*

Coal	Actual		Relative	
	F.R.L. Method, Grindability Index	Bureau of Mines Method, Lb. per Hr.[a]	F.R.L.	Bureau of Mines
Pennsylvania anthracite	74	1.33	23	30
Bellingham	177	1.56	55	36
Utah	192	1.76	60	40
Roslyn 3	198	1.86	61	43
Bayne	205	2.19	64	50
Wellington	244	2.47	76	57
Wilkeson 4	248	2.67	77	61
Wilkeson 7	322	4.37	100	100

[a] Rate of grinding in pounds of product per hour, 80 per cent of which will pass 200 mesh.

The F.R.L. index for the Pennsylvania anthracite seems low. It may be possible that the anthracite, which is so low in volatile matter as to indicate its approach to a graphite, actually lubricates the pebbles and walls of the jar mill so that slippage and consequent loss in grinding action occur. Examination of the pebbles after the conclusion of the tests seemed to add weight to this opinion. The F.R.L. procedure specifies three cycles or stages, each of 1000 revolutions. The procedure was continued on the anthracite sample for four additional stages of 1000 revolutions each, or to a total of 7000 revolutions in an attempt to explain the low value. The grindability indexes obtained for the fourth, fifth, sixth, and seventh stages were 67, 63, 59 and 59 respectively. It appears that at the conclusion of the sixth stage a stabilized condition of the circulating load is reached, but there is a continued decrease in the index up to this point.

FUTURE WORK

The method proposed in this report is presented only as a tentative procedure. Further work along this line of investigation is being continued by the Bureau of Mines. Such work may develop a better procedure. At Seattle the same coals are being used in an investigation of the equilibrium method. Such a method has been devised for ore by Maxson, Cadena, and Bond.[16] The procedure, however, as at present used in this laboratory, does not involve the maintenance of a constant circulating load; instead the test is conducted exactly as that described here, except that fresh feed is added to replace the finished product removed at the end of each cycle. The work done so far indicates a surprisingly uniform concordance with the method proposed here. The idea of an equilibrium method is now considered to be far more attractive in theory than in its practical suitability for general laboratory use.

SUMMARY

This report presents a study of various factors pertinent to the development of a standard method for estimating the ease or difficulty with which various coals are ground to the fine size required for their use as pulverized fuel. A tentative method is proposed for estimating the grindability of different coals which, it is believed, permits the elimination of more errors than is possible with other published methods. The method has been applied to eight coals covering a wide range in resistance to grinding and has been compared with a method developed by the Fuel Research Laboratories of Canada.

ACKNOWLEDGMENTS

The work described in this report was done under the direction of A. C. Fieldner, chief engineer, Experiment Stations Division, U. S. Bureau of Mines. Both he and W. H. Coghill, supervising engineer, Ore Dressing Section, Metallurgical Division, U.S. Bureau of Mines, followed the experiments and offered suggestions throughout the course of the work. The authors are also indebted to Prof. Milnor Roberts and Prof. Joseph Daniels, Department of Mines, University of Washington, for advice and criticism, and to Prof. Bror L. Grondal, College of Forestry, University of Washington, for the use of microscopic equipment.

DISCUSSION

(T. W. Harris, Jr. presiding)

R. M. HARDGROVE,* New York, N. Y. (written discussion).—The test work reported apparently has been carried on with utmost care; and all phases of the problem have been investigated in detail.

The difficulties experienced in the use of a smooth-walled Abbe mill check with our own as reported in 1931; as variations in results occurred which could not be accounted for except by slipping of the charge. In our work we developed the ball-bearing type, which gave consistent results. The authors' use of lifting ribs in the tube mill represents one solution of this difficulty but the mill then becomes a specialized design which should be made by one manufacturer in order to insure duplicate results.

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[16] W. L. Maxson, F. Cadena and F. C. Bond: Grindability of Various Ores. A.I.M.E. *Contribution* 25 (1933).

* Babcock & Wilcox Co.

Through the courtesy of the authors' we obtained samples of the eight coals reported in this paper and determined their grindability by the B.&W. method. Later we sent five samples, on which the grindability had been determined by the B.&W. method, to the authors who tested them. The agreement between the two methods was close, as the coals were all placed in exactly the same order by both methods and the range between the highest and lowest agreed within 2.5 per cent.

Fig. 5 shows the results plotted, using the B.&W. values as the abscissas. The authors' results, labeled "B.ofM." on the curve, were multiplied by a fixed constant of 25, which translates them to the B.&W. grindability scale with which we are all familiar. The values at the two ends of the curve then coincide with the B.&W. values. Individual points show a maximum difference, from the B.&W. values, of seven points but the average line drawn through the authors' points show a maximum deviation of only four points.

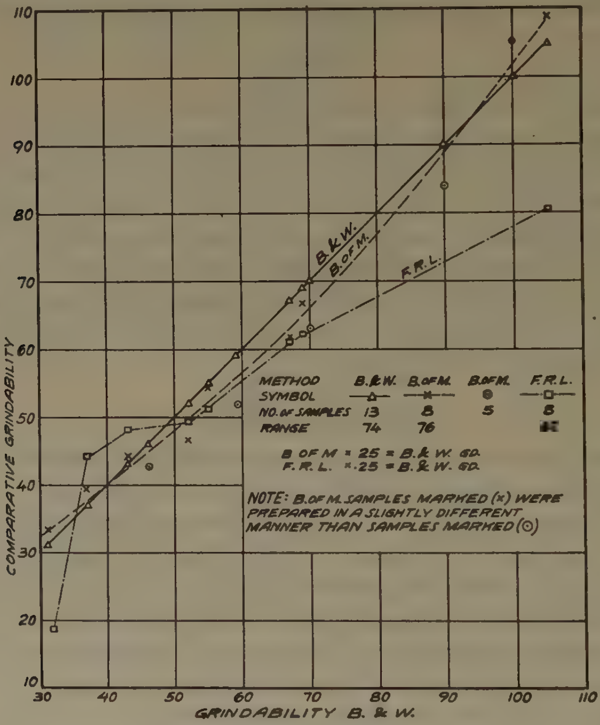

Fig. 5.—Comparison of B.&W. method of grindability determination with B.ofM. and F.R.L. methods.

The F.R.L. results, reported in the paper, when multiplied by a fixed constant of 0.25 are translated into values that agree with the B.&W. values in the vicinity of 50 grindability but show wide divergence at the upper and lower grindabilities. The range is only 62 compared to 76 for the authors' method and 74 for the B.&W.

The conclusions reached from the curve is that in the F.R.L. method the charge is cushioned with fine material at the high grindability whereas the B.ofM. method shows about the same absence of cushioning as the B.&W. method.

It is true that capacities on many present-day pulverizers correspond more nearly to the F.R.L. values, but where scavenging is performed more completely, as in closed-circuit units, with high recirculating ratios, the capacities obtained line up more closely with the B.&W. and B.of M. results. As scavenging of pulverizers is made more effective, the capacities obtained on high-grindability materials will be increased. The true grindability therefore is the one corresponding to perfect scavenging, which, while not being attained today, will be approached as the art develops.

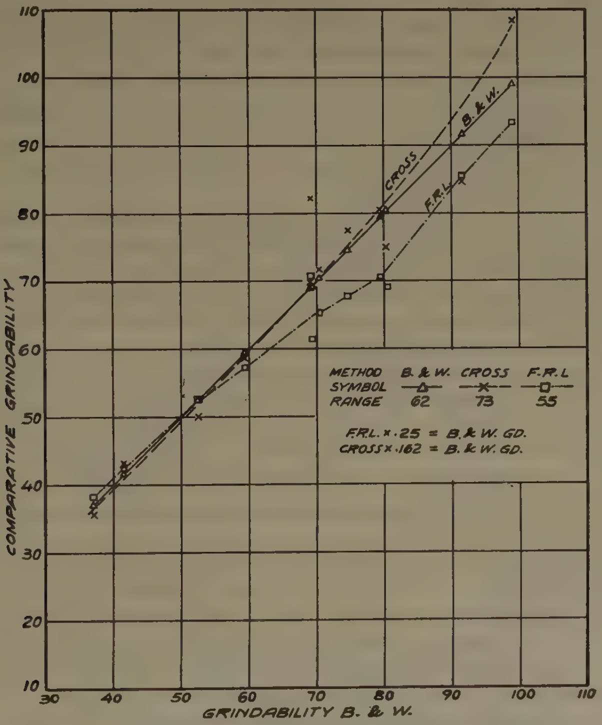

FIG. 6.—COMPARISON OF B.&W. METHOD OF GRINDABILITY DETERMINATION WITH F.R.L. AND CROSS METHODS.

In Fig. 6 are plotted in a similar way the results obtained by the F.R.L., Cross and B.&W. methods, as reported by Baltzer of the F.R.L. The relation between the F.R.L. and the B.&W. methods is similar to that reported by the authors and shown in Fig. 1, but the F.R.L. points do not show as wide a divergence at the high grindability.

The agreement between the authors' results and the B.&W. results is close and, everything else being equal, either method is a valuable determination of grindability. The time consumed, however, is far different, as the following comparison shows:

	B.&W.	B.ofM.	F.R.L.
Size of prepared sample required	50 g.	500 g.	3 lb.
Number cycles in tests	1	10–11	3
Minutes machine runs	3	15–42	–43–
Number screenings required	1	10–11	3

A test method of this kind will be much more useful if the samples required are small enough to be sent readily by mail; and the time and expense of preparing the samples and running the tests are as small as possible. The utmost attention therefore must be given to simplifying the procedure as much as possible consistent with fair accuracy.

C. E. Baltzer* and H. P. Hudson,* Ottawa, Ont. (written discussion).—Dr. Yancey and his research assistants are to be congratulated on their able presentation of the data involved as well as on the painstaking effort shown by many experiments. Their development of the new "U.S. Bureau of Mines" method merits careful consideration on the part of Subcommittee VII and all others interested in this vital question. It is noted that certain features of the F.R.L. method have been incorporated into the new method while other features have been modified to suit new viewpoints. This indicates that the F.R.L. method as originally outlined by the writers has played a part in the development of the new method and this is naturally a source of some gratification to us.

The new method, like the F.R.L. procedure, employs stage crushing of the coal and stage or "cycle" milling of 500 grams of the work sample so prepared. Whereas the F.R.L. method provides for the extraction of fines under 100-mesh fineness and the addition of new unground material after each stage of milling, the new procedure dispenses with the addition of new material but provides for much more frequent extraction of the fines under 200 mesh as the milling proceeds. This provides for much more complete scavenging of the superfines from the mill than does the F.R.L. method, and this, according to Hardgrove's conclusion as previously stated, should make the grindability results more nearly proportional to actual pulverizer capacities than these results otherwise would be. The relation between grindability and capacity is still only relative, however, and the degree of interrelation is open to some question when various types of pulverizers and the conditions under which they are operated are considered.

The new method makes no attempt to evaluate grindability by the calculation of new surface produced, as do both the Cross and Hardgrove methods, but, like the F.R.L. method, obtains the grindability value or index by more arbitrary and empirical means. This is a good feature, as results so obtained eliminate all hypothesis regarding the mean size of superfine particles, which are extremely difficult to measure with any degree of accuracy. The use of steel jars, in preference to the ordinary porcelain laboratory jars is an undesirable feature in that another piece of laboratory equipment is required to be stocked; while the ribs or "lifters" make the jar more difficult to clean than otherwise, and as more frequent cleanings are necessary with this method than with any other their use is objectionable from this standpoint. The writers have found that if the porcelain jars are efficiently loaded at least one-half full of $\frac{3}{4}$-in. steel balls the danger of breakage is reduced and more constant results are obtained between the individual determinations than if a small ball load is used. The lubricative effect of the coal is advanced as the reason for variation in the results obtained in an unribbed jar, and it is stated that in a ribbed jar "the size of the feed used in the mill has much less effect upon the resulting grindability index than when the smooth mills were used." These results might also be attributed to poor and irregular circulation of a small and relatively light load within the mill which ribbing overcomes to some extent, as does also the F.R.L. method of attack; viz., the use of an efficient mill load of approximately one-half jarful. The lengthy and carefully conducted milling procedure followed by the new method may give more precise and better coordinated results than that used by any of the other methods previously mentioned, but the many milling "cycles" and sievings through a fine-mesh sieve make the method cumbersome and subject it in a much greater degree to the same criticism that has been made of the F.R.L. procedure of stage milling in regard to "the amount of work involved and the introduction of more opportunity for discrepancies." Nevertheless, the method has many desirable features in its favor as a laboratory procedure for determining the grindability of coal.

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* Fuel Research Laboratories, Department of Mines, Canada.

C. E. Lawall,* Morgantown, W. Va. (written discussion).—This method of determining the grindability of coal will be of great interest to users of pulverized coal and to manufacturers of pulverizing machinery, because it eliminates many of the uncertain factors that are inherent in most of the other methods now used for estimating the grindability of coal.

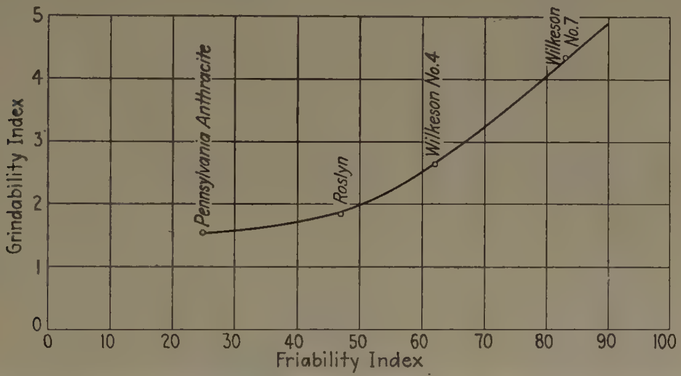

Fig. 7.—Relationship between friability and grindability indexes.

The measurement of the grindability of coal based on the energy required to crush a sample of coal to some predetermined fineness would seem to give the best results from a practical as well as a scientific point of view. The method of expressing the results of the tests as the number of revolutions required to grind a coal sample to a definite fineness; or as the rate of crushing the coal in pounds per hour is a good one because it gives the user of pulverizing machinery the information he wants.

It seems that the authors should have determined what relationship exists between the grindability of a coal and its friability. These two properties of a coal must be related to each other. In fact, the results of their grindability studies seem to indicate, in general, that such a relationship exists because the Pennsylvania anthracite coal not only is the least friable of all the coals they tested but is also the least grindable. This relationship would bear investigation because if a definite relationship could be shown between these two properties the index of friability as determined by the Bureau of Mines method might be used as a guide in many cases to estimate the grindability factor. Since the friability determination seems to be more easily and quickly determined than the grindability determination, this relationship would be very useful. Fig. 7 shows a relationship between the friability index and the grindability index of several coals tested by the authors taken from several publications, and if more coals were plotted on such a graph it might be useful in many cases as a guide for the grindability of a coal. Of course, it would be used only as a guide, because the grindability method developed by the authors is much better than the friability method for obtaining grindability data.

It is important to note that the authors found that the screen-sizing test was unreliable in evaluating the grindability index of coal. Recently a suggestion was put forward that "the surface area produced per unit of work" be used in determining the friability of coal but, as pointed out by the authors, this method may give inaccurate results, especially in the minus 400-mesh material. Microscopic counting for size estimation is the method recommended by the authors for size estimation for fine material but this method is not adaptable for a simple laboratory method.

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* Director, School of Mines, West Virginia University.

M. Frisch,* New York, N. Y. (written discussion).—Needless to say, this is an especially valuable contribution to our knowledge of the relative ease with which certain coals may be pulverized. The paper also furnishes an incentive for a great deal of speculation as to what is the best way to differentiate between coals as to the ease with which they may be pulverized and as to how to interpret such differences in terms of the performance of actual mills.

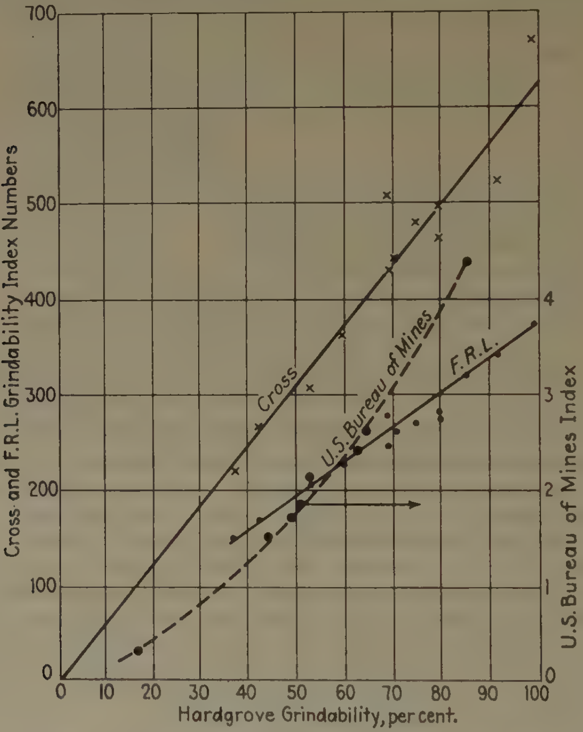

FIG. 8.—COMPARISON OF GRINDABILITY INDEXES.

The Bureau of Mines method is based on the assumption that the same amount of energy is used per revolution in treating each sample and that therefore the amount of 200-mesh produced per revolution at some stated fineness is an index of the capacity obtainable in a commercial mill. So they conclude, for example, that if Wilkeson No. 7 coal is pulverized to the same fineness as Pennsylvania anthracite the capacity obtainable with the pulverizer should be 3.29 tons of Wilkeson coal for each ton of anthracite coal pulverized, if the same amount of energy is used for pulverizing both. If this conclusion can be proved the Bureau of Mines index would be the most useful so far developed.

In order to estimate how closely this prediction checks our own experience as to the relative capacity of a pulverizer on anthracite coal and coal of the same order of pulverizability as Wilkeson coal, the relation between the F.R.L., Hardgrove and Cross indexes as given by Baltzer and Hudson[17] was used. Using this relationship the Cross, F.R.L. and Hardgrove grindability indexes will compare as shown on Fig. 8. From the known relationships between one of these indexes and the actual performance of commercial pulverizers as shown in an earlier paper,[18] it is estimated that the actual capacities at a fineness of say 70 per cent through a 200-mesh screen will be in the ratio of between 2 and 2.4 to 1 with a horizontal ball or tube mill utilizing a classifier, 3.4 to 1 with a vertical ball mill; and with a roller mill or an impact high-speed type of mill there will be an even greater difference between the relative capacities. In other words, it will be necessary with the Bureau of Mines method, as with the other methods already proposed for determining grindability, to establish an empirical relationship between the performance of a particular type of pulverizer and the laboratory index. This relationship may even have to be modified for each size of a given type, or even for the same size if changes in such factors as spring pressure, where springs are employed to create the crushing force, or size and weight of balls and speed of mill, are varied.

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* Foster Wheeler Corporation.

It is recognized already by most of the laboratory investigators and by all of the mill builders that no laboratory test seems possible which will give the relative pulverizability of all coals in all mills directly. It would seem that all tests so far proposed for determining the relative pulverizability of coals, including the proposed Bureau of Mines tests, are too complicated, require too much time and too much skill for general use in the usual run of coal laboratories such as are to be found in most of the plants using pulverized coal. Figs. 3 and 4 indicate that inasmuch as the relationship between 200-mesh product produced and the total number of revolutions is linear for all the coals up to a total number of revolutions of the mill of about 400, it would be just as descriptive of the relative pulverizability of the various coals to report the percentage of minus-200 product produced in one cycle of a total number of 400 revolutions. The index numbers so obtained would be directly proportional to the index numbers of the more elaborate and lengthy procedure of the proposed Bureau of Mines method.

By following such a procedure in our own laboratory with samples submitted to us for grindability tests in connection with operating installations and new propositions, we have been able to get just as reliable indexes of the relative pulverizability of the coals represented by the samples as we could by carrying through the Hardgrove and Cross tests. This was determined by running checks employing the three methods. In this connection it is important to note, however, that the preliminary preparation of the sample is very important. In the original Cross method the minus-200-mesh material was not rejected before the sample was treated in the grindability test machine. This resulted in inconsistencies and errors, especially with softer coals, which could be eliminated by using for the grindability test only the plus-30 minus-16 mesh as proposed by Hardgrove. We have found that by modifying the Cross method in this manner, results as consistent could be obtained with it as with the Hardgrove method.

It is my opinion that a grindability test carried out in a special jar mill with lifters as proposed by the authors on a 200 or 500-gram sample prepared as proposed by the authors and tested for a total of 400 revolutions would prove an adequate test, which can be quickly made and easily reproduced in any laboratory using the same procedure, design of mill, ball charge and speed of mill. The index could be the percentage passing a 200-mesh screen for each sample treated in this manner. If desired, some other screen could be used, as long as the particular screen used by everyone adopting this test were standardized.

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[17] Reference of footnote 2.

[18] Reference of footnote 4, Fig. 5.

C. M. Smith,* Urbana, Ill. (written discussion).—The authors of this paper have faced squarely the seemingly insurmountable difficulties that arise when an attempt is made to express the grindability of coal in terms of new surface formed. The enormous amount of new surface in the finer sizes is so dominant and yet so indeterminate as to overshadow everything else, and we have the estimated behavior of a small percentage of the test material determining almost the entire result. It becomes a case of the tail wagging the dog. For this reason, the authors are to be commended for selecting a criterion of grindability that seems to be determinable with suitable accuracy and to have possibilities of ready correlation with commercial practice.

While the apparatus and technique described by the authors are simple, and are well described, the means by which the number of revolutions required to grind 10 per cent of the sample through 200 mesh is not clear. Fig. 3 indicates that this number varied from about 50 revolutions for the Wilkeson 7 coal to 240 for Pennsylvania anthracite. Perhaps it must be determined by a preliminary run, although the authors do not indicate that such a step is necessary.

The close approach to uniformity of the successive increments of minus-200-mesh coal shown in the third column of Table 2 is a little surprising, as one would expect these increments to decrease successively as the softer material was discarded. Table 4 shows this to have been the case to a much greater extent than in Table 2, the last increments in Table 4 being roughly one-half the first.

As to reproducibility, one wishes that more data had been included, as only one pair of parallel tests, those on the Bellingham coal, is represented in Table 4. Certainly, if all parallel tests agree as closely as did these two, there can be no question about the consistency of check results to be expected from this method.

The authors' argument for the frequent withdrawal of fines seems to be cogent, as such withdrawals evidently should yield a more nearly uniform minus-200-mesh product than continuous grinding. Test data on this point would be of value.

The suggested elimination of all minus-200-mesh material from the feed, before beginning to test, would seem to be a logical necessity, for the result of a given test hinges on two things: (1) the withdrawal from the test jar of a predetermined amount of minus-200-mesh coal and (2) the number of revolutions transpiring between the beginning and the end of the test. Obviously, any minus-200-mesh coal charged with the feed requires no grinding and appears as an unwarranted credit to the grindability of the coal; in other words, it "salts" the sample. In the interest of strict accuracy, it would seem that the percentages of minus-200-mesh coal listed in Table 7 should be subtracted from all observed percentages of minus-200-mesh coal in the test data, and the number of revolutions required to produce the specified amount through 200 mesh increased accordingly. Fig. 3 indicates that this would have the effect of decreasing the reported grindability of the coals by about 5 to 10 per cent.

However, a puzzling thing about this is that the first increments of Tables 2 and 4 do not seem to reflect the presence of about 2 per cent of minus-200-mesh coal in the feed. If the first grinding produced its share of new minus-200-mesh coal, the first increment should be greater than the second by at least the amount of minus-200-mesh feed. This is not true in any of the test data given, and it would be interesting to know why.

H. C. Porter,† Philadelphia, Pa.—This is an important step in the right direction, but might it not be better to use a 5 per cent rather than a 10 per cent increment? Have any studies been made on the −200-mesh coal to detect differences that might exist in the subsieve sizes of the different coals? There are probably great differences in the new surfaces developed in the subsieve sizes of material.

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* Research Associate, Department of Mining Engineering, University of Illinois.

† Consulting Chemical Engineer.

H. F. Yancey, O. L. Furse and R. A. Blackburn (written discussion).—The complete and thorough discussions of this paper are gratifying and helpful to the authors. Professor Lawall points out the relationship between friability and grindability and shows its importance. That such a relationship exists is shown by his plot of our data for grindability and friability values. We have only determined the grindabilities of a few coals by the new method and hence are not yet prepared to present adequate data on this point. The scanty data available show that the most friable coals are the easiest to grind and this, of course, is confirmed by experience at pulverizing-coal plants. We also have at least one example of an exception to this general rule; this coal resists breakage by impact but is very susceptible to abrasion and hence easy to pulverize.

Mr. Frisch has interpreted a little freely our statement about the Pennsylvania anthracite requiring 3.29 times as much energy to pulverize as Wilkeson 7. We carefully avoided any statement or suggestion that would connect the relative grindabilities of the two coals with the capacities of commercial mills. Mill capacities depend upon so many variables that it would be idle to suggest or infer a direct relationship between laboratory grindabilities and the capacities of different mills.

It is incorrect to conclude, as Mr. Frisch does, that, because a linear relationship exists between the minus 200-mesh product produced and the total number of revolutions (Fig. 3), one cycle of 400 revolutions would be just as descriptive of the grindabilities of various coals as the method proposed here. Before advancing the grindability method described in this paper we tried, among other things, a one-cycle treatment of 500 revolutions, with the unsatisfactory results shown in Table 10.

Table 10.—*Comparison of Grindabilities by Proposed Method and a Single-cycle 500-revolution Method*

Coal	Actual		Relative	
	Bureau of Mines, Lb. per Hr.	500 Revolutions,[a] through 200 Mesh, Per Cent	Bureau of Mines, Lb. per Hr.	500 Revolutions,[a] through 200 Mesh, Per Cent
Bellingham	1.56	9.3	36	29
Utah	1.76	9.9	40	31
Roslyn 3	1.86	14.0	43	44
Bayne	2.19	13.0	50	41
Wellington	2.47	19.5	57	62
Wilkeson 4	2.67	17.6	61	56
Wilkeson 7	4.37	31.6	100	100

[a] Average of four tests on 4 to 6-mesh feed.

This gives a rough scale of relative grindabilities, but two of the coals, Bayne and Wellington, are changed from the order in which they are placed by the proposed method as well as by the Hardgrove and F.R.L. methods.

Professor Smith raises a number of interesting points. The number of revolutions required to give 10 per cent of finished product is determined by experiment, as stated in the report. Either a preliminary trial may be made, the products discarded, and the test started on a new sample, or a partial-cycle run, yielding less than 10 per cent of product, may be made. The number of revolutions for a partial-cycle run will be determined largely by the operator's experience in preparing the 10-mesh sample. He will then grind the 500-gram sample for 50, 100, or 150 turns and find the amount of through 200-mesh material produced. The number of revolutions then required to give 10 per cent of the finished product is calculated, and the through 200-mesh material and oversize is given the required number of additional revolutions to complete the first cycle. If more than 10 per cent of product results from the first run, a new feed sample should be used for the first cycle. Table 11 gives check tests on all the coals showing the reproducibility of the method, as requested by Professor Smith. Excellent agreement is shown.

TABLE 11.—*Check Determinations of Grindability by Proposed Method*

Coal	No. of Revolutions Required to Give 80 Per Cent Passing 200 Mesh	Rate, Lb. per Hr.	Coal	No. of Revolutions Required to Give 80 Per Cent Passing 200 Mesh	Rate, Lb. per Hr.
Wilkeson 7	605[a]	4.37	Roslyn 3	1425	1.86
	600[b]	4.41		1440	1.84
	620[c]	4.27			
			Utah	1500	1.76
Wilkeson 4	990	2.67		1500[e]	1.76
	1010	2.62			
			Bellingham	1700[f]	1.56
Wellington	1070	2.47		1720[g]	1.54
	1080[d]	2.45		1720[g]	1.54
Bayne	1210	2.19	Pa. anthracite	1990	1.33
	1190[d]	2.22		2005[d]	1.32
				2040	1.30

[a] 75-revolution cycle; feed sample contained 12.2 per cent ash.
[b] 75-revolution cycle; feed sample contained 10.3 per cent ash.
[c] 100-revolution cycle; feed sample contained 10.3 per cent ash.
[d] Duplicate test run by different person.
[e] Duplicate test run concurrently with first test.
[f] Feed sample contained 14.5 per cent ash.
[g] Feed sample contained 14.0 per cent ash.

Tests of feeds with and without the accompanying 200-mesh material have shown that it is unnecessary to remove the through 200-mesh from the 500-gram feed sample as long as the amounts of this material are as small as shown in Table 7, but as a precaution against overcrushing the feed sample, we have adopted and recommend the practice of using 500 grams of 10 to 200-mesh feed.

Professor Smith wishes to know why the increments of finished product produced by the first cycle are not greater than those of the second cycle by the amount of through 200-mesh material originally present in the feed. The reason this is not the case is due to the difference in the average size composition of the original feed and the material ground in the second cycle. During the first cycle the average grain size is reduced sufficiently over that of the original feed to permit the production of approximately the same amount of finished product in the second cycle as in the first cycle.

Until a shorter method, equally or more accurate, is proposed, the authors are not in sympathy with the criticisms characterizing the method as complicated and cumbersome. If a method of reasonable accuracy is desired by the industry, the expenditure of five to ten dollars for a grindability determination is small compared with the value of the information obtained. The amount involved is no more than for determination of calorific value. The total time required to run one determination is 4 to 5 hr., hence a method requiring less time but equal or better in accuracy would be very desirable.

TABLE 12.—*Comparative Grindabilities of Five Coals by Proposed Method and Hardgrove Method*

Coal	To Give 80 Per Cent through 200 Mesh		Relative	
	Revolutions	Rate,[a] Lb. per Hr.	Bureau	Hardgrove
Jerome mine, Upper Kittanning bed, Somerset Co., Pa.	630	4.20	100	100
Adrian mine, Lower Freeport bed, Jefferson Co., Pa.	790	3.55	80	90
Unknown	1050	2.52	60	70
St. Charles, Mich.	1270	2.08	50	59
Bonny Blue mine, Blue Diamond bed, Lee Co., Va.	1540	1.72	41	46

[a] Pounds per hour of finished product, 80 per cent of which is finer than 200 mesh.

Mr. Hardgrove has shown graphically a comparison of grindability values for 13 coals determined by his method and by our method. Eight of these coals are those used for our paper; the other five were furnished by Mr. Hardgrove. The values for the first set of eight coals agree well, but the other set of five coals, as given in Table 12, do not show satisfactory agreement, although the relative order is the same.

TABLE 13.—*Comparison of Proposed and Equilibrium Methods for Estimating the Grindability of Coal*

	Actual		Relative	
	Proposed Method, Lb. per Hr.	Equilibrium Method, Lb. per Hr.	Proposed	Equilibrium
Pennsylvania anthracite	1.33	1.16	30	30
Bellingham	1.56	1.41	36	36
Utah	1.76	1.59	40	41
Roslyn 3	1.86	1.62	43	42
Bayne	2.19	1.83	50	47
Wellington	2.47	2.24	57	58
Wilkeson 4	2.67	2.29	61	59
Wilkeson 7	4.37	4.04	100	104[a]

[a] Anthracite was considered as 30 in the case of the equilibrium method, which accounts for the plus-100 value of the Wilkeson 7 coal.

The reason for the lack of agreement in the second set probably is due to differences in the size of the subsieve material for the different coals.

The proposed method has been compared with the so-called "equilibrium" method mentioned in the text of the paper. The procedure used for the equilibrium method is identical with that described in the paper for the proposed method, except that fresh feed is added at the end of each cycle to replace the finished product removed by sieving. Thus a constant load of 500 grams is maintained in the mill throughout the test period. The equilibrium procedure, however, requires about 15 or more cycles compared with only 8 for the proposed method. The end point of the equilibrium method is reached when the rate of grinding or production of finished product per cycle and its ash content becomes constant. The results obtained by these two methods are compared in Table 13.

The equilibrium method requires too much time for use as a regular laboratory procedure, but it simulates actual pulverizing-plant practice as closely as can be done readily in the laboratory. The excellent agreement between the two methods indicates that the proposed method gives accurate, useful results.

Relationship between Round and Square-hole Screens for Coal*

By H. F. Yancey† and R. E. Zane,‡ Seattle, Wash.

(New York Meeting, February, 1933)

Although usually coal is screened with punched plate screens having round-hole perforations, there has been an increasing use of wire-cloth screens with square apertures, especially at pneumatic coal-cleaning plants and at other plants that use vibrating screens. It is therefore desirable to establish a relationship between round-hole and square-hole screens for use not only in experimental investigations of coal-preparation problems, but also for use in commercial operations. This need is apparent whenever it is necessary to substitute wire cloth for punched plate and still retain the familiar trade-size product.

Although similar work probably has been done by industrial organizations concerned with the screen sizing of coal, the only recorded study of the subject other than the one described herein of which the authors are aware is that of Roesler,[1] who screen-sized two samples of limestone and two of zinc concentrates crushed to pass ½ in. with a set of square and round-hole screens and found the ratio to be 1:1.23.

Procedure

In the course of a study of various methods for determining the friability of coal,[2] it was necessary to screen-size the products of a series of friability tests of three different coals with both round-hole and square-hole screens. The coals used were from the Roslyn No. 3 and the Bayne mines in Washington and the Rolapp mine in Utah. Twenty 25-lb. samples of each of these three coals at 3 to 2½ in. round-hole size were dropped from a height of 10 ft. to a concrete floor, and then screened with

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[1] H. A. Roesler: Size of Products from Square and Round-Hole Screens. *Eng. & Min. Jnl.* (1915) **99,** 493.

[2] H. F. Yancey and R. E. Zane: Comparison of Methods for Determining the Friability of Coal. U. S. Bur. Mines *Rept. of Investigations* 3215 (1933).

2½, 2, 1½, 1, and ½-in. round-hole screens and with 2, 1½, 1.05, 0.525, and 0.371-in. square-hole screens.

RESULTS

Table 1 shows the average of 20 screen analyses of each coal with both the round and square-hole screens. Fig. 1 shows graphically the data of Table 1, wherein the cumulative weight percentages of the round-hole screening tests are plotted against the diameters of the round-hole screens expressed in millimeters, for convenience in reading the scale. The screen analysis of each coal with round-hole screens is shown by the circular points and the curves connecting them. The cumulative weight percentages retained on the square-hole screens are indicated on the corresponding curves by short vertical lines. The corresponding values of the latter points on the horizontal axis represent the diameters of round-hole screens that would retain the same percentages as did the square-hole screens. In other words, these intercepts give the diameters of the round-hole screens that would retain the same amount of material as was retained on the square-hole screens. Column 10 of Table 1 shows the values of each of these intercepts on the horizontal axis. In order to establish the relationship between round and square-hole screens, it is then only necessary to divide the values for the intercepts by the opening of the corresponding square-hole screen on which the product was retained, as shown in column 5 of Table 1. These ratios are shown in column 11 of Table 1. They vary from a minimum of 1.20 to a maximum of 1.28. The average ratio obtained was 1.23. This means, for example, that a given square-hole screen will retain the same percentage of coal, similar in shape of particle to those examined, as a round-

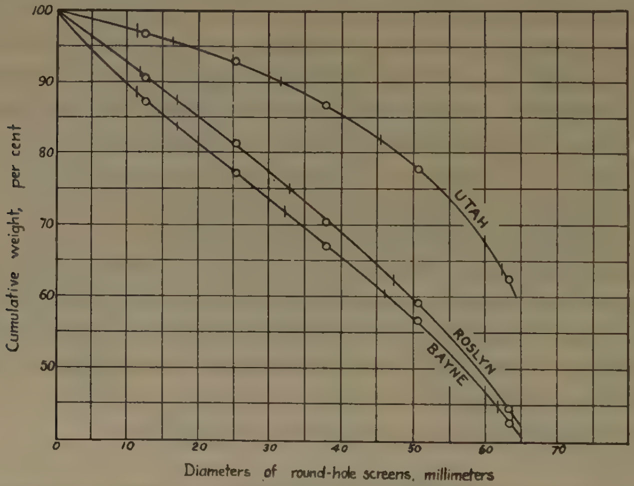


FIG. 1.—SCREENING TESTS OF THREE COALS WITH ROUND-HOLE SCREENS, SHOWING EQUIVALENT SIZE OF SQUARE-HOLE OPENINGS.

TABLE 1.—*Relationship of Round and Square-hole Screens Obtained by Screening Tests on Three Coals*

Coal	Size				Round-hole Screens		Square-hole Screens		Equivalent Round-hole Screen, Mm.	Ratio of Round to Square Hole
	Inches		Millimeters		Weight, Per Cent	Cumul. Weight, Per Cent	Weight, Per Cent	Cumul. Weight, Per Cent		
	Through	On	Through	On						
1	2	3	4	5	6	7	8	9	10	11
Utah		2.5		63.5	62.5	62.5				
	2.5	2.0	63.5	50.8	15.2	77.7	63.9	63.9	62.6	1.23
	2.0	1.5	50.8	38.1	9.0	86.7	18.0	81.9	45.7	1.20
	1.5	1.0	38.1	25.4	6.2	92.9				
	1.0	0.5	25.4	12.7	3.7	96.6				
	0.5		12.7		3.4	100.0				
	1.5	1.05	38.1	26.67			8.0	89.9	32.5	1.22
	1.05	0.525	26.67	13.33			5.7	95.6	16.5	1.24
	0.525	0.371	13.33	9.423			1.4	97.0	11.5	1.22
	0.371		9.423				3.0	100.0		
Roslyn		2.5		63.5	44.5	44.5				
	2.5	2.0	63.5	50.8	14.7	59.2	43.8	43.8	63.8	1.26
	2.0	1.5	50.8	38.1	11.2	70.4	18.6	62.4	47.5	1.25
	1.5	1.0	38.1	25.4	11.0	81.4				
	1.0	0.5	25.4	12.7	9.1	90.5				
	0.5		12.7		9.5	100.0				
	1.5	1.05	38.1	26.67			12.8	75.2	32.8	1.23
	1.05	0.525	26.67	13.33			12.1	87.3	17.0	1.28
	0.525	0.371	13.33	9.423			3.8	91.1	11.8	1.25
	0.371		9.423				8.9	100.0		
Bayne		2.5		63.5	42.5	42.5				
	2.5	2.0	63.5	50.8	14.1	56.6	44.7	44.7	61.8	1.22
	2.0	1.5	50.8	38.1	10.4	67.0	16.2	60.9	46.3	1.22
	1.5	1.0	38.1	25.4	10.2	77.2				
	1.0	0.5	25.4	12.7	9.9	87.1				
	0.5		12.7		12.9	100.0				
	1.5	1.05	38.1	26.67			10.9	71.8	32.3	1.21
	1.05	0.525	26.67	13.33			11.8	83.6	17.0	1.28
	0.525	0.371	13.33	9.423			4.8	88.4	11.4	1.21
	0.371		9.423				11.6	100.0		
Average ratio...										1.23

hole screen of which the diameter is 1.23 times the side of the square-hole opening. This relationship for coal is the same as that previously found by Roesler for limestone and zinc concentrates.

SUMMARY

A relationship between round and square-hole screens has been established by screening tests of 20 samples of each of three different coals, using a series of both types of screens. The average ratio of round to square-hole screens obtained was 1.23 to 1. The reciprocal of 1.23 is 0.81, which means that a square-hole screen with an opening 0.81 as large as the diameter of a round-hole screen will retain the same percentage of broken coal.

DISCUSSION

(*Thomas G. Fear presiding*)

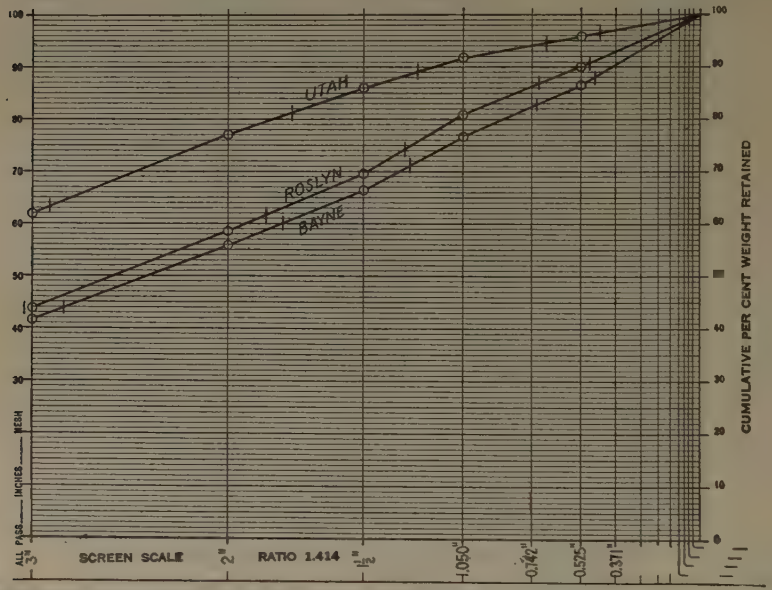
G. R. Delamater,* Cleveland, Ohio (written discussion).—It would be well if some definite standard could be established that would assure all work of this kind being conducted in like manner. It is of great interest to note that the relationship between round-hole and square-hole screens for the screening of the three coals

The Tyler Standard Screen Scale

Form No. L-4
Please mention above when ordering

Cumulative Direct Diagram of Screen Analysis on Sample of________________

Name________________ Date________________



Indicate the Screen Crushed through and also First Retaining Screen	SCREEN SCALE RATIO 1.414							ROUND HOLE SCREENS			SQUARE HOLE SCREENS					
	Openings		Mesh	Diameter Wire Inches	Sample Weights	Per Cent	Per Cent Cumulative Weights	WT. PER CENT		CUM. WT. PER CENT	WT. PER CENT		CUM. WT. PER CENT			
	Inches	Millimeters														
THROUGH																
RETAINED ON	3.000	76.20		0.207				62.5		62.5						
" "	2.000	50.80		0.192				15.2		77.7	63.9		63.9			
" "	1.500	38.10		0.148				9.0		86.7	18.0		81.9			
" "	1.050	26.67		0.148				6.2		92.9	8.0		89.9			
" "	0.742	18.85		0.135												
" "	0.525	13.33		0.105				3.7		96.6						
" "	0.371	9.423		0.092				3.4		100.0						
" "	0.263	6.680	3	0.070												
" "	0.185	4.699	4	0.065												
" "	0.131	3.327	6	0.036												
" "	0.093	2.362	8	0.032												
" "	0.065	1.651	10	0.035												
" "	0.046	1.168	14	0.025												
" "	0.0328	0.833	20	0.0172												
" "	0.0232	0.589	28	0.0125												
" "	0.0164	0.417	35	0.0122												
" "	0.0116	0.295	48	0.0092												
Pass	0.0116	0.295	48	0.0092												
				Totals,												

IN PLOTTING SQUARE ROOT OF TWO SERIES USE EVERY OTHER VERTICAL LINE

THE W. S. TYLER COMPANY, CLEVELAND, OHIO

Fig. 2.

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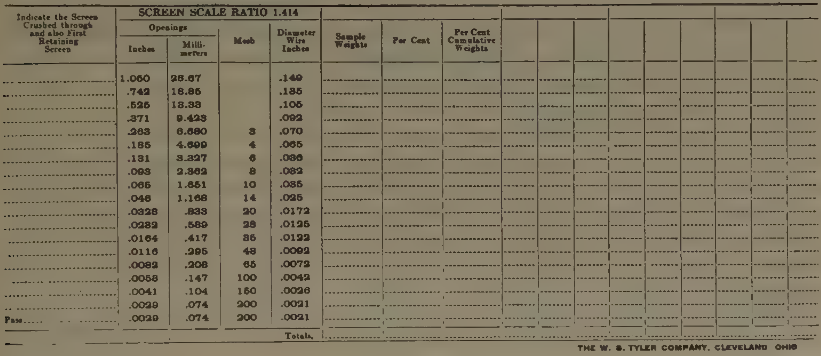
* Assistant Vice President, The W. S. Tyler Company.

tested is the same as that for limestone and zinc concentrates as determined by Roesler. It would be well if the Bureau of Mines would carry this work further by similar tests on coals of varying character and from different coal fields of the United States. I have concrete evidence of the need of such data through information recently sent me by the fuel engineer of one of the largest coal companies of the Middle West, who has but recently undertaken testing similar to that of Yancey and Zane. He says:

"We are working on the design of a chart that will be especially adapted to our purpose. . . . For a number of years, in the coal industry, it has been general practice to prepare several sizes using round-hole perforations in flat plates. . . . In recent years vibrating screens with square-mesh cloth are coming into constantly increasing use. . . . Because of the long acquaintance with round-hole screens the use of square mesh . . . will be confusing to many. . . . There is great need of a convenient and quick way to determine the exact relationship between round-hole and square-mesh screening."



Indicate the Screen Crushed through and also First Retaining Screen	SCREEN SCALE RATIO 1.414				Sample Weights	Per Cent	Per Cent Cumulative Weights
	Openings		Mesh	Diameter Wire Inches			
	Inches	Millimeters					
	1.050	26.67		.148			
	.742	18.85		.135			
	.525	13.33		.105			
	.371	9.423		.092			
	.263	6.680	3	.070			
	.185	4.699	4	.065			
	.131	3.327	6	.036			
	.093	2.362	8	.032			
	.065	1.651	10	.035			
	.046	1.168	14	.025			
	.0328	.833	20	.0172			
	.0232	.589	28	.0125			
	.0164	.417	35	.0122			
	.0116	.295	48	.0092			
	.0082	.208	65	.0072			
	.0058	.147	100	.0042			
	.0041	.104	150	.0026			
	.0029	.074	200	.0021			
Pass	.0029	.074	200	.0021			
				Totals.			

THE W. S. TYLER COMPANY, CLEVELAND OHIO

FIG. 2*a*.

TABLE 2

Round-hole Diameter, In.	Area	
	This Perforation, Sq. In.	Double Area of Next Smaller Perforation, Sq. In.
3	7.069	6.284
2	3.142	3.534
1.5	1.767	1.732
1.05	0.8659	0.865
0.742	0.4324	0.433
0.525	0.2165	0.215
0.371	0.1075	

The method of plotting the chart employed by this engineer is practically the same as that of Yancey and Zane, except that he used the Tyler standard screen scale and cumulative direct diagram chart, which needs no detailed description here, as both of these have been almost universally adopted as standard by industries engaged in screening work, and this very fact of the rather general familiarity with both screen scale and chart is a fairly good reason for careful consideration of this method as a standard for charting coal data of the character under discussion.

I also find that the National Sand and Gravel Association, Inc., Washington, D. C., has issued a similar chart for this purpose in that industry, though it has not adopted the Tyler standard screen scale, using instead fractions of an inch such as, ¼, ½, ¾, 1, etc.

In Table 1, it will be noted that the round-hole and square-mesh screens used by the authors resulted in a little overlapping of the intermediate sieves, which might be a little confusing to many, and it would seem best to avoid such condition.

Fig. 2 shows the Tyler standard screen scale chart with some alterations, and is offered as a possible standard for this work. The data in the table at the bottom of this chart are taken from the Yancey and Zane Table 1, but only for the Utah coal, and it would be possible to provide columns in this table for at least two more kinds of coal, all of which could be charted on the diagram. The curves plotted on this chart are for the Utah, the Roslyn and the Bayne coals of Table 1. The table from a Tyler Standard chart is reproduced in Fig. 2*a* for comparison.

Notice especially the substitution in my table of a 3-in. screen for the 2½-in. screen in Table 1, this being done only to illustrate the screen openings that I would recommend be used, as they fall within the Tyler standard screen scale with its fixed ratio of 1.414, or the square root of 2, which, by the way, was originally recommended by Rittinger and which provides a screen scale in which the area or surface of each successive opening is just double that of the next finer or half that of the next coarser sieve.

If thought necessary, the table on such a chart could be lengthened to take in the finer meshes to and including the 200-mesh sieves, as shown on the standard Tyler screen scale.

The use of round-hole perforations in the testing sieves, with a diameter the same as the width of the clear opening of the square-mesh cloth of the Tyler screen scale, results in extremely close adherence to the 1.414 ratio of that scale, as is evidenced by the figures given in Table 2.

R. P. Hudson,* Wayland, Ky. (written discussion).—This study of this problem seems to be a pioneer, as far as bituminous coal is concerned.

It is necessary now for coal operators to produce several sizes of coal, instead of just the three or four standard grades that were made a few years ago. Coals for use in domestic and industrial stokers must, necessarily, be closely sized. The screening of coal therefore has assumed a new and greater importance in the bituminous coal industry.

The ratio in area between a square perforation and a circular perforation, of similar diameters, is 1.2732:1. A square perforation therefore will pass 1.27 times as much material as a circular perforation of similar diameter. The reciprocal of this value is 0.7854, which means that a square perforation with an opening 0.785 as large as that of a circular perforation will retain the same percentage of material being screened. These values would hold good in actual screening practice if the various particles of the material screened were of similar shape and dimension. We know, of course, that this is not so, particularly in bituminous coal. The only method of obtaining the exact relationship is by direct experimental tests, which the authors have done, in the right way. It is probably just a coincidence that they duplicated the results of Roesler, on limestone and zinc concentrates (1:1.23).

While the authors have presented a contribution to the literature of coal preparation of considerable importance (of greater importance than is generally realized), it is regrettable that the experimental work was not based on coals having a greater commercial importance and a more extended use. Similar studies should be made on coals from the various provinces and regions. It would also be well to determine the relationship between screens of various types, such as screens containing rectangular perforations and the so-called lip screen. Such studies should be encouraged because the data obtained will assume a new and greater importance in connection with the movement now under way to standardize the screening and grading of coal at the mine.

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* Chemist, Elk Horn Coal Corporation.

J. R. Campbell, Pittsburgh, Pa. (written discussion).—In an attempt to verify the figure of 1.23 as determined by Yancey and Zane, or to establish another ratio applying to coal from this (Pittsburgh) district, screen-sizing tests were obtained on a sample of Monongahela River coal (Table 3). These tests were made on 4, 2½, 1 and ⅜-in. round-hole screens and the same sizes of square-hole screens. The data were handled in the same way as those of the Bureau of Mines. A sizing curve (Fig. 3) on the round-hole basis was constructed and square-hole cumulative percentages marked on the curve. Round-hole screen sizes were then read which would retain the same weight percentages as the square holes used. The ratios, as thus determined, were as follows: at 4 in., 1.14; at 2½ in., 1.16; at 1 in., 1.19; at ⅜ in., 1.17; average, 1.16 ratio.

Table 3.—*Size Analyses on Sample Taken from Feed Belt, Feb. 2, 1933, Over One Shift*

Pittsburgh Seam, Monongahela River Coal

Size	Square Hole		Round Hole	
	Per Cent	Cumulative	Per Cent	Cumulative
Plus 4 in.	0.31	0.31	3.26	3.26
4 to 2½ in.	12.13	12.44	14.19	17.45
2½ to 1 in.	31.05	43.49	30.60	48.05
1 to ⅜ in.	24.65	68.14	23.65	71.70
⅜ in. to 20 mesh	24.58	92.72	21.03	92.73
20 mesh to 0	7.28	100.00	7.27	100.00
	100.00		100.00	

Diameter Round Hole	Equivalent Square Hole	Ratio
4.00	4.575	1.14
2.50	2.900	1.16
1.00	1.190	1.19
0.375	0.440	1.17
	Average	1.16 Ratio Round Hole to Equivalent Square Hole

Although this ratio differs from that obtained by Yancey and Zane on western coals, the results obtained so far (we expect to carry our investigations further) check closely with each other, the greatest variation from the average being 0.03. This leads us to believe that Pittsburgh seam coals have a different fracture from that of western coals which affects the ratio of round to equivalent square-hole screens. It would be interesting to determine this ratio using other eastern coals.

A. C. Fieldner,* Washington, D. C.—This paper is a byproduct of an investigation on methods for determining the friability of coal. The discussion indicates the need of more experiments on different kinds of coals.

E. A. Holbrook.†—I want to call attention to U.S. Bureau of Mines *Bulletin* 234, issued in 1923, on screening, round versus square holes, of which I was a coauthor. We showed that ores or coal vary as they tend to be cubical, spherical or flat. We found a different ratio for each, and that there was no standard to fit them all. By experiment we found that we had to use a theoretical standard.

H. F. Yancey and R. E. Zane (written discussion).—The suggestions of Delamater and Hudson, and the data of Campbell should assist in developing a standard practice for approximating round and square-hole ratios. Campbell's ratio of 1.16 for one coal from the Pittsburgh district is considerably lower than our value of 1.23 for the three coals we tested. Thus, Campbell's lower ratio means that the diameter of the round hole and the side of the square would be more nearly the same.

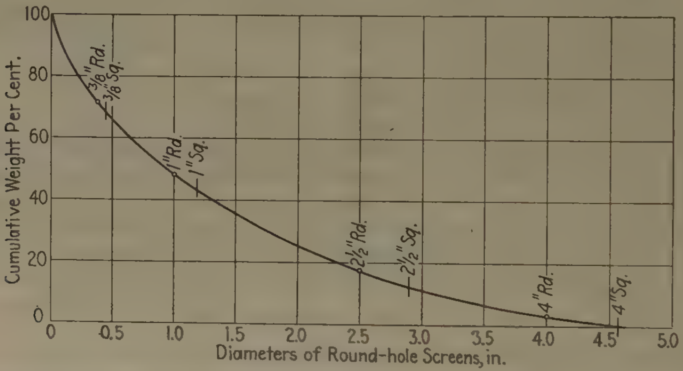

Fig. 3.—Size ratio, Pittsburgh seam coal, with round and square-holes creens.

He should supplement the one screening test of the Pittsburgh coal with several others on the same coal, using more screens, such as the scale suggested in Delamater's discussion, so as to enable a more exact curve to be drawn. Likewise, as has been pointed out elsewhere in the discussion, more work on other coals is very desirable.

Holbrook has referred to *Bulletin* 234 of the Bureau of Mines. In this report he and Fraser state that a 1-in. square hole will produce approximately the same amount as a 1¼-in. round-hole screen; thus the ratio of round to square is 1.25. Although no data are given, Holbrook and Fraser conclude that in comparing round and square holes of the same rated size the areas are as 0.7854 to 1, and the respective amounts of material are in almost the same ratio. This means that round and square holes of equal areas would pass the same amount, or that the ratio of the round-hole diameter to the square-hole side would be 1.13.

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* Chief Engineer, Experiment Stations Division.

† Dean, School of Engineering and Mines, University of Pittsburgh.

Reactivity of Anthracite with Carbon Dioxide

By W. L. Keene,* H. G. Turner† and G. S. Scott,‡ State College, Pa.

(New York Meeting, February, 1934)

It is well known that different forms of carbon show differences in chemical behavior under apparently identical conditions. It is fairly well known that these differences persist, although to a lesser degree, in a single group of carbons, such as the anthracites of Pennsylvania. It is the purpose of this paper to show the extent of these variations for Pennsylvania anthracites from actual quantitative measurements of the amount of carbon dioxide reduced under the standardized test conditions employed. As the amount of carbon dioxide reduced to the monoxide (under comparative conditions termed, for convenience, the "reactivity") is important in combustion, gas production and metallurgy, it is hoped that this paper will be a useful contribution to the store of knowledge on these subjects.

In carrying out the investigation, considerable time and thought were spent in obtaining what seemed to be the best possible design of apparatus. The next step consisted in making a number of trial runs on the same sample, varying temperature, particle diameter, etc. and then outlining a standardized procedure to be followed in testing all samples. Fifty anthracites were tested. As a matter of interest, some 10 miscellaneous samples were also tested.

The data show initial reaction temperatures (abbreviated I.R.T.), reaction velocity constants at 900° and 950° C., some effects of added substances, and a few additional tests designed to throw some light on the causes of differences in reactivity.

The sample of charcoal, as was anticipated, proved to be easily the most reactive of any sample tested. The anthracites as a group came second. The five samples of gas and by-product cokes, obtained from five different coke manufacturers, came third, and a single sample of Rhode Island anthracite ranked fourth, with respect to reactivity as above defined. Although the five coke samples do not necessarily represent the average market coke, and five samples constitute a rather small number of samples to serve as a basis for comparisons, the fact that none of the coke samples was as reactive as the least reactive anthracite in a group of 50 would seem to be sufficient evidence for questioning the idea, prevalent in some quarters, that coke is more reactive than anthracite.

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* Metallurgist.

† Director of Research for the Anthracite Institute, The Pennsylvania State College.

‡ Assistant Director of Research; Anthracite Institute, Department of Research, The Pennsylvania State College.

Previous Investigations

It is not hard to understand why investigators so frequently disagree in reactivity studies. The very term itself has many different interpretations. To quote a few examples: Bunte[1] defines the reactivity of coke in terms of its ignition temperature in air under a set of fixed conditions. Fisher, Breuer and Broche[2] passed CO_2 over coke that was being heated at a constant rate and determined the point at which CO appeared, taking this as a measure of the reactivity. Sherman and Kinney[3] streamed oxygen over a predetermined weight of coke and took the weight loss after a given time as a measure of the reactivity. Another definition appeared when Rieffel[4] used the term "reducing power." (This definition the writers have accepted as satisfactory for their purpose.) Similar to this is the empirical definition of Jones et al.,[5] who expressed reactivity values as the number of milliliters of CO formed from 100 ml. of CO_2 at 950° C.

One of the first investigators of the reaction of CO_2 with carbon was Boudouard[6] who determined more or less accurately the equilibrium of CO_2 with various forms of carbon.

Clement, Adams and Haskins[7] studied the rate of formation of CO from CO_2 and carbon at temperatures between 900° and 1300° C. They tested a coke, a charcoal and an anthracite and came to the conclusion that at the lower temperatures the differences in the reaction rates were more noticeable, the reaction of coke being slowest and that of charcoal the fastest. At higher temperatures and lower gas velocities, the percentage of CO formed was practically the same for each of the three forms of carbons.

In discussing the work of these investigators, Lewis and Randall[8] state that the work of Boudouard gave the order of magnitude of the equilibrium constants only. However, this work was superseded by the careful measurements of Rhead and Wheeler,[9] who, besides obtaining the variation of the percentage CO and the equilibrium constant with temperature and pressure, established the fact that the reaction $C + CO_2 = 2CO$

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[1] Bunte: *Brennstoff Chem.* (1923) **4**, 167.

[2] Fisher, Breuer and Broche: *Brennstoff Chem.* (1923) **4**, 33.

[3] Sherman and Kinney: *Fuel* (1926) **5**, 98.

[4] Rieffel: *Chem. u. Ind.* (1931) **26**, 2, 285.

[5] Jones, Sinnatt and King: *Jnl.* Iron and Steel Inst. (1928) **117**.

[6] Boudouard: *Compt. rend.* (1899) **128**, 824.

[7] Clement, Adams and Haskins: U. S. Bur. Mines *Bull.* 7 (1911).

[8] Lewis and Randall: Thermodynamics. New York, 1924. McGraw-Hill Book Co.

[9] Rhead and Wheeler: *Jnl.* Chem. Soc. (1910) **97**, 2178; (1911) **99**, 1141.

proceeds many times faster to the right than to the left (at 850° C., 166 times). This follows from their equilibrium constants. At the same time, they concluded that the reaction is essentially one of surface phenomena.

Lewis and Randall also state that in the work of Clement et al. the rate of reaction was chiefly studied, and that their estimate of the equilibrium constant was evidently erroneous, as the experiments were of too short duration to permit attainment of equilibrium conditions.

Rieffel, of Essen, developed a method and an apparatus that embodied many advantages apparently not generally recognized. These points will be brought out in the description of the writers' apparatus, which is a modification of that used by Rieffel.

Mechanism of the Reaction

The character of the carbon surface is one of the most important factors in the reduction of the carbon dioxide. Amorphous forms, such as charcoal, are relatively highly reactive, while the more stable and inert form, graphite, is but slightly reactive.

In the reaction $C + CO_2 = 2CO$, the rate of diffusion of CO_2 to the carbon surface at fuel-bed temperatures is greater than the velocity of the chemical combination, therefore chemical reaction rate rather than diffusion is the controlling factor. On the other hand, the velocity of the chemical reaction $C + O_2 = CO_2$ at fuel-bed temperatures is so great that the diffusion of oxygen to the carbon surface is slower than the chemical reaction and therefore is the factor controlling the speed with which the reaction proceeds. In most chemical reactions (at room temperature) the rate in general doubles for each 10° C. rise in temperature. The increase in velocity in most of the combustion reactions, however, is far less, since the increase in rate of reaction for a given increase in temperature falls off markedly at higher temperatures. At temperatures around 1000° C. it usually requires an increase of 100° C. to double the rate.

The rate of reaction may be expressed by the equation $k_1 = \frac{dC}{dt}$ (that is, the change in concentration in unit time if the concentration of the reacting substance is unity). If one of the substances is a solid, k_1 depends on the nature and the amount of the solid surface.

According to Fox and White[10] no acceptable mechanism has yet been proposed for the reduction of a stream of carbon dioxide by solid heated carbon, which will permit calculations such as are possible in the case of ordinary combustion. They state that attempts to apply diffusion or fluid-flow calculations to this case are rendered difficult by the fact that the volume of CO flowing from the interface is double the volume of CO_2 approaching the interface, and that therefore there is a general drift away from the carbon surface, within the relatively stationary surface film. The thickness of the layer is not constant and the calculation of the transport of material through it is very difficult.

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[10] Fox and White: *Ind. & Eng. Chem.* (1931) **23**, 264.

Concerning the differences in reactivity existing in the various modifications of carbon, Dent and Cobb[11] contend that coke consists of various substances or constituents with different reactivities. These substances have an effect on carbon dioxide at different temperatures, and the reactivity of a coke at one temperature is likely to be very different from that at another temperature. Several writers[12] agree that exposing a coke to a high temperature for a period of time definitely lowers its reactivity toward CO_2. This fact has also been observed in the present investigation. Parr and Staley[13] state that for each advance in temperature carbon assumes a definite form, which directly affects its reactivity.

APPARATUS

The apparatus used in this investigation is similar to that of Rieffel,[14] who had previously reviewed many different methods of testing reactivity. During the actual construction of the apparatus, however, some alterations were made which appeared advantageous, and a new method of determining the initial reaction temperature of carbon with carbon dioxide was devised. A sketch of the completed apparatus is shown in Fig. 1, and is believed to be nearly self-explanatory. A few additional notes may aid in clarification.

It was necessary to determine the percentage of CO produced by passing CO_2 through the heated fuel bed, and in order to eliminate a complicated gas analysis, pure CO_2 from the tank was used as the source of supply. The gas passed from the tank through a Sargent wet-test meter, and then through an H_2SO_4 drying bottle. A manometer was cut into the line to measure delivery pressure. A gas reservoir, of which the function will be explained later, was next in the series, followed by a flowmeter, with which a rough estimate of the flow could be obtained. From this, the gas either could pass into the first gas burette for measurement or could be bypassed. It was then free to pass into a silica tube, which was suspended in a vertical electric combustion furnace, the temperature of which was measured by several thermocouples and

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[11] Dent and Cobb: *Colliery Guardian* (1929) **138**, 2208.

[12] Reference of footnote 5.
Adge: Fuel Economist (1928) **3**, 243.
Sensicle: Gas World (1930) **92**, 13.

[13] Parr and Staley: *Fuel* (1928) **7**, 540.

[14] Reference of footnote 4.

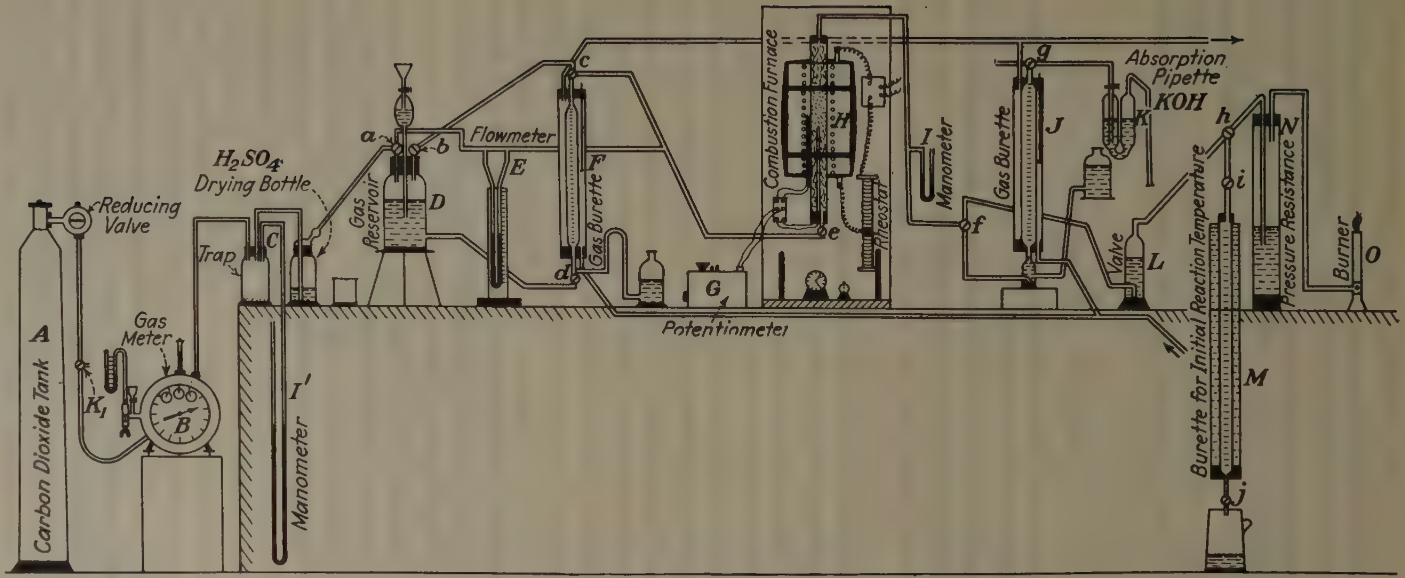

Fig. 1.—Apparatus for study of reactivity of anthracite versus other solid fuels, Anthracite Institute.

regulated by a rheostat. Another manometer measured the pressure on the right-hand side of the furnace. After passing through the fuel bed, the gas was turned in either of two ways. During the sampling, it passed up through acidified salt water into the second water-jacketed gas burette for measurement and analysis. When not sampling, it passed through a mercury valve *L*, the function of which was to maintain a constant back-pressure, thence to an auxiliary pressure adjuster *N*, and to the burner *O*. Part *M* is a long, graduated burette used in the determination of the initial reaction temperature.

Method of Operation

A selected sample of anthracite or coke was crushed, thoroughly mixed and screened to the desired size, then dried at 105°C. Exactly 50 c.c. was measured into a graduated cylinder, weighed and placed in the combustion tube on a bed of broken silica. An iron-constantan thermocouple was placed so that the hot junction was in the middle of the fuel bed.

The CO_2 gas was then turned through the system and all air flushed out, the fuel being thoroughly saturated. The stopcock *e* at the bottom of the sealed silica tube was then closed, and the right-hand side of the system connected to *M*, the large burette. The furnace was then heated at a definite rate and the temperature recorded against time. The volume of gas accumulating in *M* was read at 10-c.c. intervals against time. At the initial reaction temperature the gas accumulated more rapidly, owing to the reaction of one volume of CO_2 with carbon to produce two volumes of CO. Although it would take an appreciable time interval for the CO formed to pass into *M*, the increased rate was noticeable instantly in the flow rate, and a subsequent plot of the data determined accurately at what temperature this sudden expansion took place. This temperature was designated as the initial reaction temperature and in itself is an indication of the reactivity of the carbon.

When the temperature at which the reducing power was to be measured had been attained, the rheostat was set to maintain this temperature. The gas was then swept through the system until a steady state was attained, the waste gas being burned in burner *O*. The flow was adjusted to a predetermined rate and samples taken for analysis, as follows: gas was led into the top of the gas reservoir by closing stopcock *b* and opening *a*, meanwhile reversing *f*. The gas pressure on the liquid in *D* forced the gas out of the burette *F* and through the furnace, to be collected in the second burette *J*, and measured. After a sample was taken (normally 30 sec. was allowed) the gas was again sent through the bypass system and burned at *O*. As the gas in burette *J* now consisted only of CO_2 and CO, running it several times through the absorption pipette *K* (filled with 40 per cent KOH) removed the unreduced CO_2, and the remaining CO was measured. After becoming essentially constant, this procedure was repeated for each sample at least six times. The data from these tests were used as the basis of calculation, thus eliminating the effect of the salt water on the gas composition. That this procedure was satisfactory is shown by the fact that check runs agreeing to within 1 per cent reducing power were consistently obtained with normal care in operation.

METHOD OF CALCULATING RESULTS

Rate of Flow.—During the calibration of the apparatus, test results showed that the rate of flow could best be calculated from the gas volumes in the second burette. The flowmeter and the timed rate in the first burette served as admirable checks on this calculation. Half-minute sampling periods were used and from the amounts of CO_2 and CO, the rate of flow was obtained by adding, to the amount of CO_2 measured, half of the amount of CO measured, and multiplying by two to obtain rate per minute.

Contact Time.—In order to calculate the contact time, the porosity of the fuel bed was first calculated, using the weight, volume and specific gravity of the fuel. Contact volume and rate of flow, with temperature corrections, then gave the contact time in seconds.

Specific Reaction Rate—Constant k_1.—The formula developed by Clement, Adams and Haskins[15] was used to obtain this value. That is:

$$k_1 = \frac{2.3}{t} \log_1 \frac{1}{1 - x}$$

where t = contact time in seconds,
x = fraction CO measured.

Percentage of CO Produced.—In order to have a basis of comparison between the various fuels, the percentage of CO was calculated at a fixed temperature to an arbitrarily selected seven seconds contact time. From the previously given formula and the value of k_1 and with $t = 7$, the percentage of CO (x) was easily determined.

Reducing Power.—A formula developed from the relations between the reducing power and the percentage of CO enabled the former to be obtained easily. This formula is:

$$\text{R. P.} = \frac{x}{2 - x} \qquad \text{where } x = \frac{\%\text{CO}}{100}$$

The purpose of calculating the reducing power and the percentage of CO was for comparison of the different fuels. The specific reaction rate constant was used in determining the effect of temperature and particle size within certain limits on particular samples. It must be remembered that the fuels tested in this investigation were not pure carbons, but natural or prepared fuels whose action might be influenced by many extraneous factors. Bearing this in mind, the temperature constants were calculated for the equation $2.31 \log k_1 = \frac{A}{T} + BT + C$ from the data obtained at the two temperatures used: i. e., 900° and 950° C.; and for the third point, the value for $k_1 = 0.0001$ at the initial reaction temperature. We find this point $k_1 = 0.0001$ by assuming that the percentage of CO at the I.R.T. is 0.10 per cent. This is not strictly accurate but the error is not great.

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[15] Reference of footnote 7.

The effect of size may be calculated by the formula:

$$k = \frac{k_1}{A} \text{ and } k' = \frac{d_1}{d_2} k_1$$

where d_1 and d_2 are the respective particle diameters.

Determination of Initial Reaction Temperature

The method of obtaining the data leading to the initial reaction temperature has already been given. This procedure gives results comparable to those obtained by other investigators using different methods, and yields the true temperature at which the reaction begins. This point is also indicative of the reducing power at the higher temperatures in the case of anthracites. Fig. 2 gives a plot of the data obtained on an anthracite. This curve is typical of most of the fuels tested. The shape of the gas curve may vary according to the fuel, in some cases

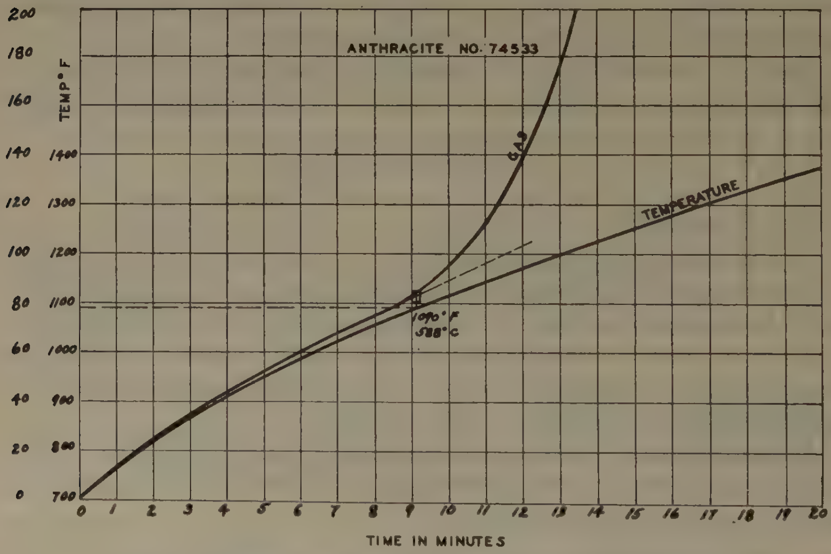

Fig. 2.—Determination of initial reaction temperature.

being flatter, and, in the case of the cokes, having a less distinct "break point." It is believed that the adsorptive power of the carbon surface for CO_2 is responsible for variations in the shape of the curve. The anthracites in every case gave off more CO_2 than did the cokes before the break point was reached.

It is interesting to note that the determination of the initial reaction temperature, unlike empirical methods for the determination of ignition temperatures of coals, is but little affected by radiation and convection errors, since the inside of the tube approximates black-body conditions.

EFFECT OF TEMPERATURE AND PARTICLE SIZE

Before testing the main body of samples, experiments on the effect of temperature and particle size were made in order to determine standard conditions for the tests.

Table 1 gives the results of tests on a selected sample of anthracite and a sample of coke at a temperature of 900° C., contact time seven seconds.

TABLE 1.—*Effect of Size on Reducing Power and* k_1

Temperature, 900° C., contact times, 7 seconds

Size, Mm.	Coke A		Anthracite A	
	k_1	R.P.	k_1	R.P.
1.41–1.00	0.0640	22.0	0.0926	31.3
2.00–1.41	0.0571	19.8	0.0889	30.7
2.38–2.00	0.0490	17.0	0.0832	28.4
4.70–2.38	0.0280	9.8	0.0724	24.8

ANALYSIS OF SAMPLES

	Volatile Matter	Ash	Fixed Carbon	Sulfur
Coke A	1.7	9.6	88.7	0.7
Anthracite A	6.4	9.3	84.3	0.7

Table 2 shows the effect of temperature on the reaction velocity or reducing power. The actual and calculated data are shown for some samples of anthracite and coke as used above, by varying the temperature from 800° to 1000° C. by 50° steps. The furnace used was not capable of sustained temperatures much above 1000° C. without rapid deterioration, but tests were run at 1075° C. at the conclusion of the experiments, and the results were substantiated. The reaction rates for the two fuels approach a common value at the higher temperatures employed, greater differences appearing at the lower temperatures. The anthracite shown here is more reactive than the coke—by chance this coke being the most

TABLE 2.—*Effect of Temperature on Reducing Power and k_1*

1.41–1.00 mm. size. 7 sec. contact time

Temperature, Deg. C.	Coke A				Anthracite A			
	k_1		R.P.		k_1		R.P.	
	Observed	Calculated	Observed	Calculated	Observed	Calculated	Observed	Calculated
1200		3.18		100.0		2.98		100.0
1150		1.54		99.9		1.55		99.9
1100		0.769		99.1		0.829		99.3
1050		0.393		88.0		0.454		91.9
1000	0.207	0.207	62.1	62.1	0.257	0.257	71.7	71.7
950	0.113	0.113	37.6	37.6	0.151	0.151	48.6	48.6
900	0.064	0.064	22.0	22.0	0.093	0.093	31.3	31.3
850	0.031	0.038	10.9	13.1	0.054	0.059	18.7	20.5
800	0.013	0.023	4.4	8.1	0.035	0.042	12.2	14.7
750		0.015		5.3		0.029		10.1
650		0.0001		0.1		0.019		6.5
550						0.0001		0.1

$A = 13842$, $B = 0.02101$, $C = -39.191$.
$A = 14915$, $B = 0.02019$, $C = -38.774$.

highly reactive of all the cokes tested subsequently. The anthracite, on the other hand, might be classed as low to medium in comparison with other anthracites. Fig. 3 shows the curve drawn from the calculated data.

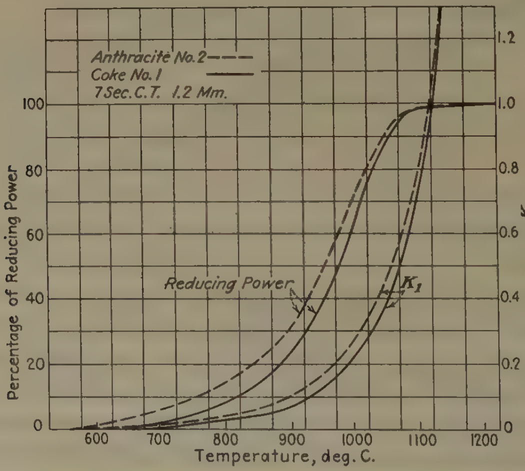

FIG. 3.—EFFECT OF TEMPERATURE ON REDUCING POWER AND K_1 (COAL AND COKE).

EFFECT OF CATALYSTS

There are several substances which, when added to carbon, increase its reactivity with CO_2 to varying degrees. Perhaps the most important of these is sodium carbonate, the effect of which has been noted by many investigators. Other substances, such as iron salts, the oxides of iron, chromium and manganese, alkalis and alkaline earth salts and lime, have a similar but not so marked effect. It has been found in this investigation also that the use of a nichrome tube in place of the silica combustion tube accelerates the reduction. Baker[16] found in his experiments with well dried gases that moisture has a catalytic effect. He found that charcoal at a bright red heat does not reduce *dry* CO_2 to the monoxide.

TABLE 3.—*Results of Temperature Tests on a 2.8 Per Cent Ash Anthracite and Same Anthracite plus 5 Per Cent* Na_2CO_3

Samples in silica tube, 7 sec. contact time.
Initial reaction temperature 590° C. and 550° respectively.

Temperature, Deg. C.	Coal Alone			Coal Plus Soda		
	k_1	R.P.	CO, Per Cent	k_1	R.P.	CO, Per Cent
700	0.0274	8.5	17.0	0.0470	16.3	28.1
800	0.0625	21.6	33.1	0.1940	59.1	74.3
900	0.1336	43.7	60.8	0.5203	94.9	97.4
1000	0.2733	74.4	85.3	0.6200	97.4	98.7
1075	0.4384	91.2	95.4			

TABLE 4.—*Results of Tests on 2.8 Per Cent Ash Anthracite with Additions of Various Catalytic Agents (in Silica Tube Except where Noted)*

Temperature 950° C., 7 sec. contact time.

	k_1	R.P.	CO, Per Cent	Increase in Per Cent CO
Untreated anthracite	0.1697	53.4	69.6	
Anthracite plus 5 per cent soda	0.6234	97.4	98.7	29.1
Anthracite plus 5 per cent lime	0.1999	60.5	75.4	5.8
Anthracite plus 5 per cent iron oxide	0.2374	68.2	81.1	11.5
Anthracite in nichrome tube	0.2412	68.9	81.6	12.0
Untreated anthracite at 1030° C.		79.0	88.0	
Anthracite plus 5 per cent lime at 1030° C.		96.5	98.2	10.2

Tables 3 and 4 give the results of tests on a low-ash anthracite with the addition of various catalytic agents. Fig. 4 is a plot of the data.

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[16] Baker: *Phil. Trans.* (1888) **179**, A571.

The addition of a small amount of sodium carbonate to a low-ash anthracite raises the reactivity values to a maximum at the temperature employed; i. e., 98.7 per cent CO at 950° C. The percentage of CO present at equilibrium for a pure carbon at 950° C. is 98.67 per cent (see footnote 8). This would indicate that equilibrium is practically obtained in seven seconds as against several hours with an untreated carbon. Fe_2O_3 and CaO have similar but lesser effects. A theory explaining the behavior of Na_2CO_3 is given in a paper by Fox and White (footnote 10).

From a study of the preliminary tests, the following conditions were chosen as standard for the main body of tests:

Particle size. 1.41 to 1.00 mm. (14 to 18 mesh U. S. standard).

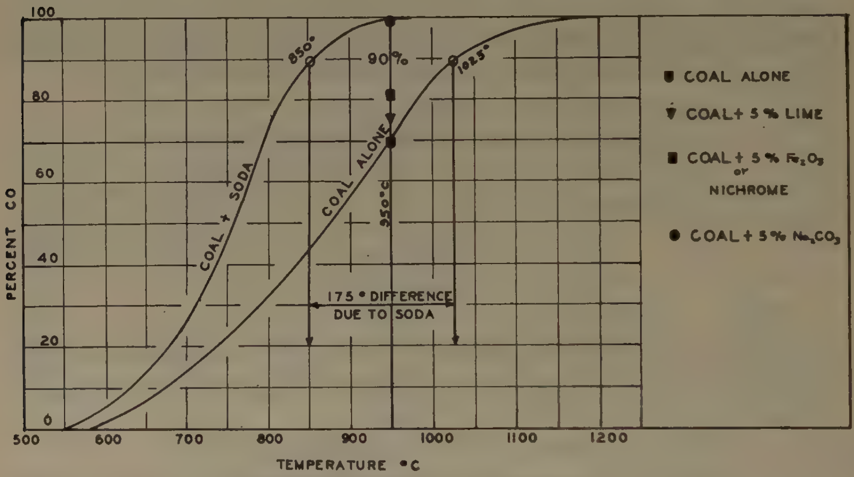

Fig. 4.—Effect of catalysts.

Temperatures. One test was made at 900° C. and another at 950° C. on each sample, in which range the temperature variation curve has the greatest slope and is nearly straight. The initial reaction temperature is determined at the beginning of each test, which affords a check on this determination.

Contact time. A rate of flow calculated to give as near seven seconds contact time as possible was chosen. The results were then calculated to exactly seven seconds.

Procedure. As outlined, except that after a sample reached the desired temperature it was held for 15 min. with gas flowing before analyzing samples. This was to remove any traces of volatile matter and adsorbed air and allow conditions to adjust themselves to a steady state.

Discussion of Results

In Table 5 is given a complete summary of the data taken on 50 anthracites, 5 cokes and 5 other modifications of carbon. The anthracites were selected from the collection of over 200 face samples taken by Professor Turner in cooperation with the U. S. Bureau of Mines and represent a complete cross-section of the types of coal existing in the anthracite fields. The samples tested were chosen so as to cover the entire range in composition with special emphasis on extremes in ash, volatile matter and specific gravity.

Fig. 5 shows a plot of reducing power against initial reaction temperature. The figures for reducing power are the averages of the two figures obtained at 900° and at 950° C. The relationship shown is decidedly marked and is interesting in that it shows about how closely one may be taken as an indication of the other.

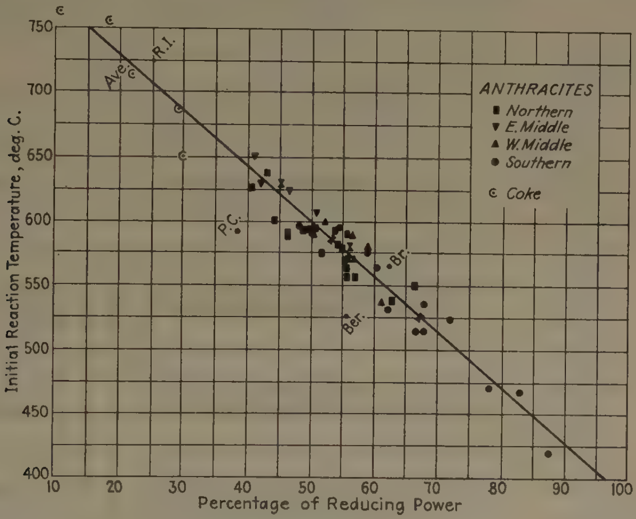

FIG. 5.—INITIAL REACTION TEMPERATURE VERSUS REDUCING POWER.

In speculations as to the reasons for variations in reactivity, when considering anthracites, most engineers with whom the writers have discussed the matter feel that the volatile matter and the specific gravity are in some way related to the reactivity, and some engineers feel quite certain that the ash exerts some influence. Fig. 6 shows the initial reaction temperatures plotted against these three properties just mentioned. With respect to volatile matter, there is a definite relationship, but from the poor correlation it seems evident that the I.R.T. is influenced decidedly by other factors as well. No correlation is evident in the relationship with ash. Against specific gravity the correlation approaches zero, but that a relation does exist will be shown later by other data.

Jones et al. (reference of footnote 5) found that the addition of as little as 0.05 per cent of iron to coke causes a marked increase in reactivity values, and that larger amounts do not increase the reactivity proportionately but do increase the permanence of the reactivity. In this

TABLE 5.—*Reactivity*

ANTHRACITE FACE SAMPLES NOT BREAKER PREPARED

No.	I.R.T.	Specific Reaction Rate Constant		CO, Per Cent, Average	R.P., Per Cent, Average	Specific Gravity	Volatile Matter	Ash	Vein
		900° C.	950° C.						
				Northern Field					
1	590	0.1374	0.1790	67.0	50.0	1.50	5.0	6.8	Top Clark
2	640	0.0928	0.1701	60.0	43.0	1.54	6.5	10.2	Bottom Clark
3	580	0.1556	0.1932	71.0	54.0	1.56	7.5	13.7	4-foot
5	580	0.1440	0.2078	71.0	55.0	1.54	6.5	10.8	14-foot
6	595	0.1160	0.1884	66.0	49.0	1.66	7.3	23.5	New County
7	565	0.1612	0.1933	71.0	55.0	1.47	6.7	5.4	Rock Top B
8	585	0.1706	0.2473	77.0	63.0	1.62	5.4	10.8	Top Red Ash
9	550	0.1976	0.2613	80.0	67.0	1.54	4.6	4.7	Bottom Red Ash
10	595	0.1189	0.1913	66.0	50.0	1.67	6.3	22.2	Red Top Ash
11	585	0.1334	0.2033	69.0	53.0	1.54	6.2	9.0	Abbott
12	570	0.1465	0.2134	72.0	56.0	1.59	7.0	13.3	Top Hillman
13	575	0.1389	0.1876	68.0	52.0	1.53	5.3	7.5	Middle Hillman
14	600	0.1093	0.1607	61.0	44.0	1.56	7.3	9.6	J Hillman
15	595	0.1423	0.2012	70.0	54.0	1.59	5.7	8.3	Ross
16	595	0.1395	0.1758	67.0	50.0	1.59	5.2	9.4	Hillman
17	595	0.1226	0.1876	66.0	50.0	1.56	5.4	6.1	Bennett
18	555	0.1415	0.2145	71.0	55.0	1.65	6.0	11.3	Bottom Red Ash
20	625	0.1042	0.1385	57.0	40.0	1.51	5.5	6.0	5-foot
21	590	0.1478	0.1144	72.0	56.0	1.62	6.9	13.6	Top Ross
22	565	0.1299	0.2266	71.0	55.0	1.59	6.7	14.8	Abbott
23	555	0.1446	0.2217	72.0	57.0	1.52	6.9	7.0	Lance
Average..................	585	0.1378	0.1989	69.0	53.0	1.57	6.2	10.7	
				Eastern Middle Field					
25	630	0.1037	0.1548	60.0	42.0	1.67	4.2	18.5	Gamma
26	610	0.1228	0.1997	68.0	51.0	1.63	3.3	6.1	Buck Mountain
28	625	0.1238	0.1668	64.0	47.0	1.67	4.2	16.7	Orchard
29	650	0.0994	0.1513	58.0	41.0	1.61	3.2	4.9	Mammoth
Average..................	630	0.1124	0.1682	63.0	45.0	1.65	3.7	11.6	

Western Middle Field									
30		600	0.1462	0.1854	69.0	52.0	1.64	4.1	12.7	Little Primrose
31		590	0.1447	0.1766	68.0	51.0	1.63	4.4	12.3	Mammoth B
32		590	0.1416	0.2228	72.0	56.0	1.76	6.9	24.4	Little Orchard
33		580	0.1630	0.2238	74.0	59.0	1.73	5.9	22.5	Little Orchard
34		570	0.1436	0.2252	73.0	57.0	1.58	3.4	7.0	Mammoth Mid. B.
36		535	0.1573	0.2479	76.0	61.0	1.56	9.3	18.6	Holmes
Average....................		580	0.1494	0.2136	72.0	56.0	1.65	5.7	16.3	
Southern Field									
38		515	0.1842	0.2856	81.0	68.0	1.67	4.6	11.4	Mammoth
39		515	0.1663	0.2887	80.0	66.0	1.63	3.6	8.9	Skidmore
40		565	0.1457	0.2525	75.0	60.0	1.68	4.7	15.5	Orchard
41		530	0.1527	0.2599	76.0	62.0	1.64	4.6	9.9	Orchard
44		575	0.1683	0.2185	74.0	59.0	1.64	5.9	10.4	Primrose
45		595	0.1340	0.2141	71.0	54.0	1.60	4.5	8.8	7-Foot
46		595	0.1184	0.1780	65.0	48.0	1.70	4.9	16.9	Buck Mountain
47		420	0.2633	0.5099	93.0	88.0	1.55	9.8	13.3	No. 5
48		525	0.1918	0.3265	84.0	72.0	1.51	8.2	10.6	Holmes
49		535	0.1717	0.2977	81.0	68.0	1.58	8.6	19.4	Orchard
50		470	0.2458	0.3503	88.0	78.0	1.48	8.5	10.3	Big Lykens
51		465	0.2505	0.4251	91.0	83.0	1.48	8.7	7.4	Big Lykens
Average....................		525	0.1827	0.3006	80.0	67.0	1.60	6.4	11.9	

TABLE 5.—(*Continued*)

SUMMARY OF AVERAGES OF THE ANTHRACITES

Field	I.R.T.	Specific Reaction Rate Constant		CO, Per Cent, Average	R.P., Per Cent, Average	Specific Gravity	Volatile Matter	Ash	Fe_2O_3 in Ash
		900° C.	950° C.						
Northern	585	0.1378	0.1989	69.0	53.0	1.57	6.2	10.7	8.51
E. Middle	630	0.1124	0.1682	63.0	45.0	1.65	3.7	11.6	5.11
W. Middle	580	0.1494	0.2136	72.0	56.0	1.65	5.7	16.3	8.51
Southern	525	0.1827	0.3006	80.0	67.0	1.60	6.4	11.9	7.59
Grand average	572			72.0	57.0	1.60	6.0	11.9	
				Cokes					
No.									Description
1	695	0.0656	0.1036	45.0	29.0				Gas coke
2	765	0.0245	0.0393	20.0	11.0		0.3	10.8	Byproduct coke
3	695	0.0409	0.0806	35.0	21.0		1.1	6.5	Gas coke
4	770	0.0356	0.0685	31.0	18.0				Byproduct coke
5	650	0.0664	0.1099	46.0	30.0		1.7	9.6	Gas coke
Average	715	0.0466	0.0804	35.0	22.0		1.0	9.0	
				Miscellaneous					
Charcoal	265	0.4218	0.6080	97.0	95.0				
Coal Shale		0.0413	0.0709	32.0	19.0	2.67			
Rhode Island Anthracite	725	0.0452	0.1010	40.0	25.0	2.49		40.0	Graphite
Petroleum Coke	590	0.1080	0.1200	55.0	38.0		3.3	0.4	"C 99"
Hudson Special	590	0.1215	0.1614	63.0	46.0	1.53	6.8	2.8	Cleaned anthracite
Bernice	525	0.1341	0.2235	72.0	56.0	1.46	8.2	10.8	Semianthracite
Brady Silt	565	0.1777	0.2390	77.0	62.0	1.57		10.0	Western middle field
Composite	560	0.1631	0.1973	72.0	56.0	1.58		11.2	Mixture of all anthracites

series of experiments total iron determinations were made on all samples. For the southern field there was a fairly good correlation between total iron content and reactivity, but this relationship did not hold for the other fields. Since iron oxide has been shown to act as a catalyst, it is possible that the form of the iron may be the deciding factor. A discussion of this subject may be found in the work of Jones cited above.

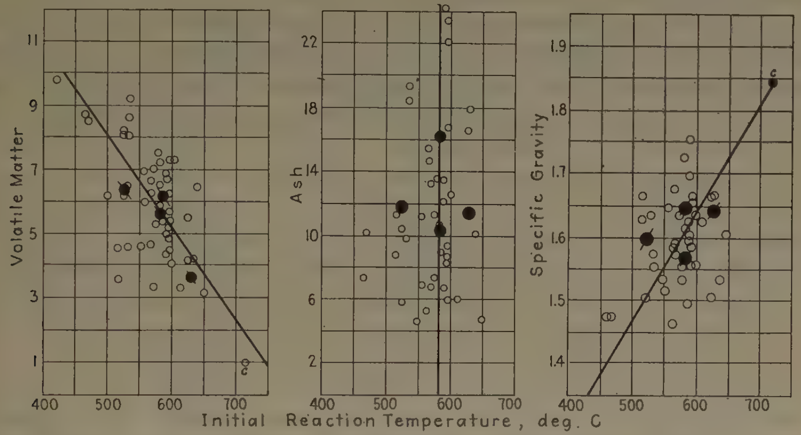

FIG. 6.—INITIAL REACTION TEMPERATURE VERSUS VOLATILE MATTER, ASH AND SPECIFIC GRAVITY.

Table 6 contains data obtained in the attempt to learn more about the causes for differences in the reactivity of anthracites. The samples consisted of composites representing the four fields, and one very reactive anthracite marked *C*. The term "de-ashed" appearing in this table

TABLE 6.—*Results of De-ashing and De-volatilizing Experiments*

Composite Number	A-1	A-2	A-3	A-4	B-1	B-2	B-3	B-4	C
I.R.T. original, deg. C	580	630	570	530	570	625	525	500	420
I.R.T. de-ashed, deg. C	620	655	620	635	615	653	610	630	595
I.R.T. de-ashed and de-volatilized, deg. C	645	665	635	655	640	660	635	655	635
Specific gravity	1.57	1.64	1.63	1.57	1.57	1.66	1.65	1.60	1.55
Volatile matter, per cent	7.3	4.2	5.3	6.5	6.3	3.6	6.2	6.2	9.8
Ash, original, per cent	11.2	6.0	14.8	9.9	11.5	8.9	15.4	10.4	11.5
Ash, residual, per cent	1.0	2.3	1.8	1.8	1.4	2.9	2.0	1.8	2.5
Volatile effect, deg. C	25	10	15	20	25	7	25	25	40
Ash effect, deg. C	40	25	50	105	45	28	85	130	175
Ash analysis per cent:									
SiO_2	54.4	52.2	54.4	49.9	56.5	53.1	55.5	51.6	40.3
TiO_2	1.8	2.2	3.6	4.0	1.2	2.0	2.0	2.0	1.3
Al_2O_3	31.1	33.2	24.9	31.3	36.0	34.8	30.3	28.2	20.4
Fe_2O_3	10.1	7.4	11.0	8.7	3.8	3.7	6.6	7.0	18.7
CaO	0.3	0.9	0.6	0.6	0.8	1.6	1.6	3.3	8.9
MgO	0	1.4	0.5	0.4	0.3	0.3	0.3	1.1	3.0
Alk	1.2	2.1	1.2	1.6	1.2	1.9	1.7	2.1	1.0
Softening temperature, deg. C	2660	2850	2660	2670	2980	2940	2840	2570	2160

means that the mineral matter of the coal has been reduced by hydrofluoric and hydrochloric acids to the quantity shown as "per cent ash, residual." The term "de-volatilized," means that the sample has been heated to remove the volatile matter. The specific gravity was determined against water after boiling 15 minutes.

In Fig. 7 is shown the rise in I.R.T. due to the removal of the volatile matter. The diameters of the circles around the points are equal to the writers' estimation of the error of measurement. This curve shows very definitely the relation between volatile content and I.R.T., which, when other influencing factors are present, was shown only as a trend (see again Fig. 6).

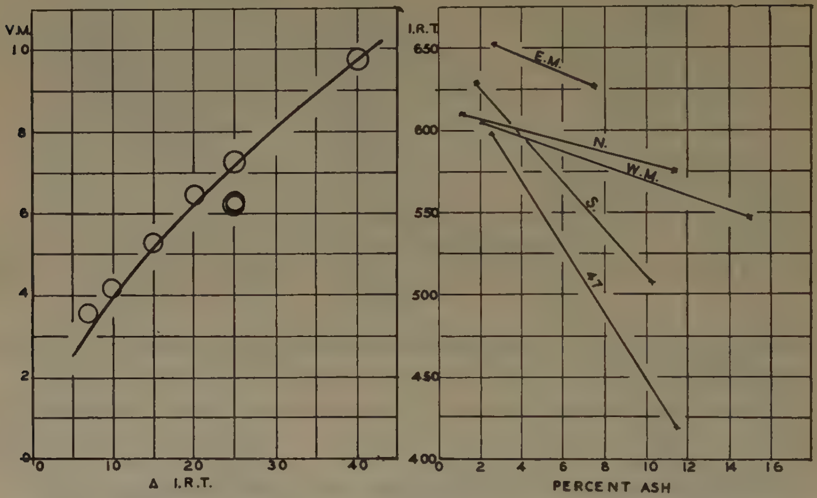

Fig. 7.—(*a*) Effect of volatile matter on initial reaction temperature of de-ashed coal; (*b*) effect of ash on initial reaction temperature of composites.

Fig. 7 also shows definitely that removal of ash by chemical means brings about an increase in I.R.T. This correlates with the effect of catalysts and is strong evidence that the ash has a catalytic effect on the reactivity of anthracites. The quantitative effects of the various constituents of the ash on the reactivity are difficult to determine. The chemical compositions of the ashes are shown in Table 6, together with the other data, so that any readers so inclined may not be denied the pleasure of searching for the relationships.

Another interesting relationship not shown in the present paper is a close correlation between I.R.T. and the specific gravity of the carbonaceous aggregate; that is, the carbon itself plus the capillary spaces created throughout the carbon by removal of volatile matter and mineral constituents. Data from Table 6 form the basis for this statement.

CONCLUSIONS

The average reducing power of the anthracites between 900° and 950° C. is 57 per cent, with an initial reaction temperature of 572° C. The average reducing power of five coke samples under the same conditions is 22 per cent, with an initial reaction temperature of 715° C.

The main factors influencing reactivity of anthracites are: ash composition, volatile matter and specific gravity of the carbonaceous aggregate. There are also many other contributing factors.

Sodium carbonate has a strong catalytic effect on the reactivity of carbons in general, while iron oxide and lime have appreciable but lesser effects.

Explanations have been attempted for the inherent differences existent in coals from different parts of the anthracite region, and for different forms of carbon.

ACKNOWLEDGMENT

The writers wish to acknowledge the cooperation of the Mineral Industries Experiment Station of The Pennsylvania State College.

DISCUSSION

(H. J. Rose presiding)

E. L. BULLER,* Scranton, Pa.—What is the relation between reaction temperature and ignition temperature in anthracite?

W. L. KEENE.—They are two different things, the latter being dependent on the size of particles, the rate of heating, the type of apparatus and other variables while the former, reaction temperature, varies little with external condition. It is a chemical property of the carbon.

E. L. BULLER.—Might not the composition of the ash affect analysis of anthracite? This is illustrated by the fact that addition of sodium carbonate promotes fluxing, thereby affecting the fusion point of the ash.

E. L. KEENE.—Fox and White have shown that under such conditions the carbonate would yield sodium, which would be volatilized but would not lower the fusion temperature of the ash. It is not a question of one constituent. Some tests in which the fusion temperature of the ash was determined electrically brought out no mathematical relationship between that temperature and the reactivity.

H. H. LOWRY,† Pittsburgh, Pa.—What is the effect of greater velocity on the reducing power? Diffusion is perhaps as important as simple reactivity, inasmuch as it is necessary to get the reacting gas to the carbon, and the products away from it. With a given specific rate of chemical reactivity at the surface the rate of reaction has been shown to vary within a fourfold range by changing the gas velocity.

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* Combustion Engineer, Hudson Coal Co.

† Director, Coal Research Laboratory, Carnegie Institute of Technology.

W. L. Keene.—We have not yet thoroughly investigated the effect of differing the rate of diffusion on the reactivity, although with CO_2 it is controlled by the lowest rate of diffusion.

H. H. Lowry.—While the rate of the chemical reaction at low temperatures may be the controlling factor, this rate is approximately doubled for each 10° (Centigrade) increase in temperature. The temperature coefficient of the diffusion process is very much less. We may assume, therefore, that at blast-furnace operating temperatures the over-all rate of reaction is dependent wholly on the diffusion of the gases to and away from the incandescent carbon, the reaction itself being instantaneous. The differences in reactivity of charcoal, anthracite and coke probably do not affect blast-furnace operations.

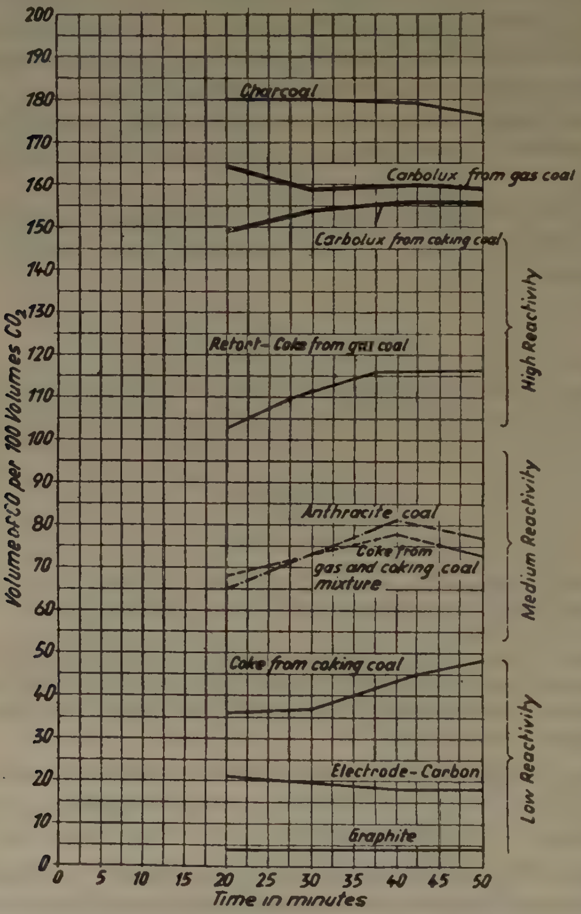

Fig. 8.—Reactivity of carbolux, anthracite, charcoal, coke, etc. (Reproduced from *Colliery Guardian*, Oct. 20, 1933.)

J. B. Morrow,* Pittsburgh, Pa. (written discussion).—As a further illustration of the well-known idea that "the difference of opinion is what makes a horse race interesting," I am quoting from an article by Dr. Heinrich Koppers.[17] As a part of his article Dr. Koppers presents a chart (reproduced in Fig. 8), showing reactivity of low-temperature coke, anthracite, charcoal, coke, etc.

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* Preparation Manager, Pittsburgh Coal Co.

Dr. Koppers goes on to state that, "The reactivity is determined according to a method developed in the laboratory of Heinrich Koppers, Ltd., Essen, by passing carbon dioxide at a fixed rate of flow over a sample of the fuel crushed to a certain size (0.5 to 1 mm = 1⁄50 in. to 1⁄25 in.), which is maintained at a temperature of 950° C. According to the reactivity of the fuel, more or less of the carbon dioxide is reduced to carbon monoxide. The result is expressed in volumes of carbon monoxide produced from 100 volumes carbon dioxide. As one molecule of carbon forms two molecules of carbon monoxide, the highest result that can be obtained is 200, and is the maximum theoretical value. In order to give some idea of the values obtained by the test, it should be realized that fuels with a reactivity index of 0 to 50 are of low reactivity, that those with an index between 50 and 100 have a medium reactivity, and those with an index between 100 and 150 have a high, while those above this figure possess a very high reactivity."

This chart rather plainly shows that it is considered in other quarters that anthracite coal is more reactive than coke made from coking coal and about equally reactive with coke made from a gas and coking-coal mixture. On the other hand, retort coke made from gas coal is definitely more reactive than anthracite and low-temperature coke is still more reactive than retort coke, and approaches closely the reactivity of charcoal.

It would be interesting if the writers of this paper would carry on further experiments with the standard method that they have developed and make tests on retort coke from gas coal and also on low-temperature coke. It is a well-known fact that the reactivity of coke can be varied according to the type of coal used in its manufacture and the temperature of burning.

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[17] H. Koppers: Recent Developments in Coking Practice. *Colliery Guardian* (Oct. 20, 1933) 724.

Application of the Micropyrometer for Determining Fusibility of Coal Ash

By Roy P. Hudson,* Wayland, Ky.

(New York Meeting, February, 1933)

Abstract

A micropyrometer known as the De Graaf apparatus has several advantages over the gas-furnace method for determining ash fusibility. When the De Graaf apparatus is operated by a modified method of procedure, the results closely check those of the gas-furnace method. This is indicated by tests on 27 samples of coal ash, with an extreme variation in fusibility values of 654° F. However, most of the ashes fused at temperatures between 2300° and 2500°. This, presumably, is the temperature interval within which the micropyrometer gives the best results. Additional comparisons are desirable.

DISCUSSION

(*Thomas G. Fear presiding*)

A. C. Fieldner,† Washington, D. C. (written discussion‡).—The Pittsburgh Experiment Station of the Bureau of Mines has made two investigations[1] of the micropyrometer for determining the fusibility of coal ash. These studies were undertaken with a view to adopting the micropyrometer method as a substitute or alternate to the present standard gas-furnace method, in view of the obvious advantages of greater speed, smaller ash sample, smaller laboratory space requirements, less equipment and greater comfort to the operator. However, the investigation showed that the micropyrometer type of furnace as then operated in five laboratories having this equipment could not be accepted as an alternate to the standard gas furnace, which is capable of close duplication in different laboratories, as shown in a recent comparison between the Pittsburgh Experiment Station of the United States Bureau of Mines and the Fuel Research Station of Great Britain.[2] The algebraic mean difference in 12 samples was 15° F. and the largest difference between the two laboratories was only 54° F.

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* Chemist, Elk Horn Coal Corporation.

† Chief Engineer, Experiment Stations Division, U. S. Bureau of Mines.

‡ Published by permission of the Director, U. S. Bureau of Mines.

[1] A. C. Fieldner, W. A. Selvig and W. L. Parker: Comparison of the Standard Gas Furnace and Micropyrometer Methods for Determining the Fusibility of Coal Ash. *Ind. & Eng. Chem.* (1922) **14**, 695.

W. A. Selvig: Check Determinations of Fusibility of Coal Ash with the DeGraaf Electric Coal Ash Fusion Furnace. U. S. Bur. Mines *Rept. of Investigations* 3003 (1930).

[2] J. G. King, A. Blackie and J. O. N. Millott: The Fusion Point of Coal Ash (Part 1), 15. *Tech. Paper* 23, Fuel Research, Dept. Sci. and Ind. Research. H. M. Stationery Office, London, 1929.

Recently the Pittsburgh Station of the United States Bureau of Mines has made an experimental study of the relation of the fusibility of ash to clinker formation in furnaces.[3] In this investigation the micropyrometer fusion tests did not show as good a correlation with the quantity of ash fused to clinker as did the standard gas-furnace method.

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[3] P. Nicholls and W. A. Selvig: Clinker Formation as Related to the Fusibility of Coal Ash. U. S. Bur. Mines *Bull.* 364 (1932).

Report of Committee on Evaluation of Coal for Blast-Furnace Coke

By Ralph H. Sweetser,* Chairman

(New York Meeting, February, 1934)

The Committee on Evaluation of Coal for Blast-furnace Coke reports progress in the working out of the problem of evaluation. There has been no full meeting of the Committee but there have been several conferences with two or more of the members present, and the different members have expressed themselves freely on the five points that have been under consideration since the meeting of the Committee in New York on Feb. 16, 1932.

The need of a method of coal evaluation, not only of coking coals but all other kinds, is obvious. Under NRA it becomes imperative that coals of the same size from the same seam be priced according to their variations in quality, especially in reference to cleanliness in respect to freedom from extraneous ash-forming materials.

In looking over the correspondence that has passed between members of our Committee, it is evident that much of the confusion and misunderstanding would disappear if we could realize that practically all of the variations in value of coal from the same seam of coal are due to variations in the amounts of extraneous materials that are contained in the coal as it passes over the scales for shipment. It is not now a question with our committee as to the differentials in value, or in prices, between different kinds of coking coals or between noncoking and coking coals; our present problem is to arrive at the method for determining the scale of values of the coal from a certain coal seam according to the increase or decrease of extraneous ash and moisture from a predetermined base line adopted for that particular coal.

The five points mentioned above are the same as discussed at the meeting in February, 1932, except that the temperature for determining the moisture content has been changed from 212° F. to 221° F., at the suggestion of Dr. A. C. Fieldner. The points are:

1. It is proposed that the coal industry adopt as a base line for evaluating all coals the analysis of the coal as it passes over the weighing device that determines the tonnage of coal bought and sold.

2. It is proposed that the Committee on Evaluation of Coal for Blast-furnace Coke approve the adoption of the base line for Pittsburgh seam coal presented by F. A. Jordan at the annual meeting of A.I.M.E.

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* Consulting Engineer.

in February, 1930; to wit: A total ash content of 9.5 per cent (at 221° F.) containing 5 per cent by weight of plus 1.55 sp. gr. material.

3. It is proposed that, in addition to the 9.5 per cent of total ash in the base line for Pittsburgh seam coal, 3.2 per cent of moisture be included, making a total of 12.7 per cent ash (determined at 212° F.) plus moisture. This is the same as 9.196 per cent ash plus 3.2 per cent moisture in the natural state.

4. It is proposed that the scope of the evaluation of coal be extended to other kind of coals besides coal for blast-furnace coke.

5. It is proposed that Pittsburgh seam coal be adopted as the base-line coal for the evaluation of all coals.

As a matter of record, and in order to emphasize the need of close cooperation of our Committee with other committees working on similar coal problems, the comments of the members of our Committee are, in whole or in part, herewith made a part of this report.

COMMENTS OF MEMBERS

A. C. Fieldner, Aug. 2, 1933: Agrees to the "five points in memorandum (Feb. 16, 1932) except that total ash content should be calculated to coal dried at 221° F., which is usual temperature prescribed in the standard methods for analyzing coal. . . . Next step in evaluation of coal is to work out a schedule of bonuses and penalties for deviation from base-line coal. (It will be difficult.) Also, should you not include some factor bearing on the physical properties of the coke produced from different coals?

"I suggest that our Committee should use the same specific gravity as recommended by the clean coal committee (American Standards Association's Sectional Committee on Specifications for Clean Bituminous Coal. H. M. Lawrence, Secretary, 29 West 39th St., New York)."

Since Dr. Fieldner is Chairman of the A.S.A. Committee on Classification of Coals, and both he and I are members of the A.I.M.E. Committee on Correlation of Research, it is all the more necessary, as well as convenient, that our recommendation in the Committee on Evaluation of Coal for Blast-furnace Coke should be in harmony with other committees working on similar problems.

I had an interview with Mr. H. M. Lawrence on Oct. 2, and learned that the specific gravity of 1.60 approved by the Sectional Committee on Specifications for Clean Bituminous Coal (American Standards Association), T. W. Guy, Chairman, at its meeting on Feb. 21, 1933, was "accepted as a basis for further work of the subcommittee." This committee agreed that: "A definition gravity for refuse should be set up, the sink at which should be considered refuse in all bituminous coal. This gravity shall be 1.60 specific gravity."

Mr. Lawrence and I agreed that there was no conflict between the specific gravity of 1.60 selected as the dividing line between clean bituminous coal and refuse, and the "plus 1.55 sp. gr." adopted by our Committee as the dividing line for "high-gravity material." The 1.60 sp. gr. applies to "refuse in all bituminous coal," and the 1.55 sp. gr. applies to "coal for blast-furnace coke."

Carl Scholz, Aug. 10, 1933: "I think I can readily agree to the five points, agreeing with Mr. Hudson on the importance of sampling. I feel that a study of the prevention of fines in mining may prove of interest and benefit. When the user takes the sample after it has been crushed, he cannot tell much of the condition of coal as it was loaded, and my thought is the separation of the mine sample into four or five sizes and a separate study of each. . . . With mechanical loading I find this all the more important. . . . The higher prices which follow the wage increases will call for improvements in quality."

K. H. Marsh: "The Department of Mines at Ottawa is making a study of the coking coals of Nova Scotia and we may be able to have some data for our evaluation committee."

Charles C. Russell, June 8, 1933: "It seems to me that one of the most important messages that could be put across is the necessity for accurate sampling of coal and coke. As long as the producer and consumer of coal for metallurgical coke can contest analyses on the basis of unsatisfactory sampling methods, coal evaluation will, I believe, be difficult to establish. In the second place, I believe that it would be very valuable for the Committee to recommend a method for the determination of the amount of slate (heavy-gravity material) in coal. I have in mind that a test of this kind should be made part of coke-oven and mine-laboratory procedure. It would be a simple matter to determine the sink at 1.60 sp. gr. in the coal to the ovens. This would mean using carbon tetrachloride only. With the daily data from this test, it would simplify the evaluation of heavy-gravity material."

In his article, "Effect of Extraneous Matter in Coal on the Physical Properties of Blast-furnace Coke" (presented at the session of the Raw Materials and Blast-furnace Committee of the Iron and Steel Division on Feb. 19 of this year), Roy P. Hudson makes a strong plea for clearing up the meaning of some of the terms used in our discussions on the evaluation of coking coals. Asking, "What is high-gravity material?" Mr. Hudson says:

"The term, 'high-gravity material,' or 'heavy-gravity impurities,' has recently come into general use in the bituminous coal industry. It is a term rather difficult to define and the specific gravity of this material varies in different coals. A more explanatory and descriptive term is needed for this substance and the author believes the term '*extraneous matter*' conveys a more accurate conception of its physical and chemical qualities. The term 'extraneous *ash*' does not accurately designate the properties of this material, because it may, or may not be wholly ash."

It is the hope of the committee that this progress report will help to clarify the difficulties of agreeing on methods of evaluation of coals for blast-furnace coke, and also all other coals. Evidently the time has come when the pioneering work of this committee should be turned over to a group in close touch with the commercial side of evaluation under the Coal Code of NRA. Effective Jan. 1, 1934, all byproduct coking coals on General Price Circular No. 9, of Appalachian Coals, Inc., were listed at the price of $1.85 per ton f.o.b. mine "to all points including Lake and Tidewater. All grades and sizes." From such a base price applied to the base coal in any particular seam, it should now be possible to build up a scale of premiums, for improved quality, equitably dividing the benefits of increased value between the producer and the consumer.

Anthracites and Semianthracites of Pennsylvania

By Homer G. Turner,* State College, Pa.

(New York Meeting, February, 1934)

The coals mined in the area known as the anthracite fields of Pennsylvania have always been recognized commercially as anthracites. In the literature, however, some of these anthracites have been called semianthracites by various writers.

The coals mined in Sullivan County, especially those from the Bernice field, have been recognized commercially both as anthracites and semianthracites. In the literature they have almost always been referred to as typical semianthracites, although Franklin Platt,[1] in his report to the Pennsylvania Geological Survey, uses the terms semibituminous, semianthracite, and anthracite in describing them.

The purpose of this paper is to endeavor to dispel the confusion of names used to describe these coals and to supply further information that will permit a logical and definite classification.

Most of the confusion of the past has been due to the impossibility of securing enough fresh face samples from which to draw sound conclusions; some of it was due, no doubt, to lack of standard methods of analysis, and of course, to the ever-present lack of definition of boundaries from the standpoint of classification.

MAP SHOWING LINES OF EQUAL VOLATILE MATTER AND LINES OF EQUAL SPECIFIC GRAVITY

As a foundation for this work the writer constructed a map showing lines of equal volatile matter and lines of equal specific gravity throughout the anthracite and semianthracite regions. The points on these lines were determined from a study of analyses of 214 face samples from 28 mines and composite analyses from 18 different breakers. These samples were collected through the joint efforts of the U. S. Bureau of Mines and the Anthracite Institute. The Bureau of Mines standard method of sampling was used in all cases. All analyses were made by the Bureau of Mines and are used here with its permission. The specific gravities were determined on the face samples in our laboratory, using a method that will be described later.

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* Director of Research for the Anthracite Institute, The Pennsylvania State College.

[1] F. Platt: Report of the Progress of the Second Geological Survey of Pennsylvania, Lycoming and Sullivan Counties, Chap. V, 1880.

The figures given on the isovols were obtained by correcting all analyses to a moisture-and-ash-free basis instead of using the 1.1 × ash-free correction[2] proposed in report No. 5 of subcommittee II of the coal classification committee. This was done because all existing isovol maps of other regions in Pennsylvania are constructed on the old moisture-and-ash-free basis, and because practically all published analyses are calculated on this basis. The lines on the map would have essentially the same positions regardless of whether they represent isovols, isocarbs, or dry 1.1 × ash-free isocarbs; the numerical values of the lines would be different.

To check this, all analyses were calculated to the dry 1.1 × ash-free basis and plotted on the map. It was found that the values of the isovols, as given, could be converted to the new basis by subtracting the volatile matter, as given, from 100 and adding 1.2. The resulting figures will be accurate to within ±0.5 per cent, which would be the maximum error. The one isocarb given on the map is exactly as calculated from the analyses.

The volatile matter as calculated on the moisture-and-ash-free basis from the face samples was given greatest weight in locating the isovols. Several points, however, were located by using the volatile matter indicated by the breaker samples. It was concluded that this latter figure was fairly reliable, for careful comparison showed that the breaker samples were practically the same as the face samples for a given mine. The average of all proximate analyses of face samples as compared to the average proximate analyses of all breaker output showed identical figures for volatile matter, 0.7 per cent difference in fixed carbon, 0.7 per cent difference in ash, and a 0.1 per cent difference in sulfur. The maximum difference in volatile between face samples and samples from breakers preparing the same coals was 0.6 per cent.

In addition to these data, analyses from various operators were used to test the position of the lines plotted.

Present Status of Anthracitic Groups

At the last meeting of the coal classification committee the tentative classification of Table 1 was proposed.

This classification, when modified slightly, fits the facts very well and seems to be the best that we can hope to obtain. "Anthracite" would be preferable to "normal anthracite." The adjective "normal" is neither descriptive nor correct. In fact, it is misleading in that it implies that meta-anthracite and semianthracite are nearer to anthracite than they really are. If an adjective is necessary, "true" would be much more fitting than "normal," for coals are no more normal than individuals. With the terms "meta-anthracite" and "semianthracite" to describe coals related to anthracite, it is unnecessary to modify anthracite to make it apply to a definite class of coal.

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[2] Dry 1.1 × ash-free basis $= \text{F.C.} \times \dfrac{100}{100 - (m + 1.1A + 0.1S)}$

Moist 1.1 × ash-free basis $= \text{F.C.} \times \dfrac{100}{100 - (1.1A + 0.1S)}$

TABLE 1.—*Tentative Classification*

Group	Class	Limits of Fixed Carbon, Mineral Matter Free	Requisite Physical Properties
1. Anthracitic	1. Meta-anthracite	Dry F.C., 98 plus	
	2. Normal anthracite	Dry F.C., 92 plus, minus 98	
	3. Semianthracite	Dry F.C., 86 plus, minus 92	Non-agglutinating

PHYSICAL PROPERTIES

The question as to whether the Pennsylvania anthracites and semianthracites can be distinguished by differences in physical properties can best be answered by listing these properties side by side as in Table 2.

TABLE 2.—*Physical Properties*

	Anthracite	Semianthracite
Luster	Vitreous to submetallic	Vitreous to almost dull
Structure	Laminated to amorphous Laminations obscure due to uniformity of luster	Laminated to amorphous Laminations very distinct due to luster contrasts
Fracture	Conchoidal to rectangular Chiefly conchoidal	Rectangular to conchoidal Chiefly rectangular
Streak	Dark brown to black Chiefly black	Dark brown to black Chiefly black
Hardness	2.75 to 3.00	2.50 to 2.75
Apparent sp. gr. with ash less than 14 per cent	1.46 to 1.70	Less than 1.47
Toughness	23 to 51 per cent friable. Average, 35 per cent friable	46 to 53 per cent friable. Average, 50 per cent friable

These properties are all averages of a great number of observations; exceptions may be expected in individual samples. This tabulation shows that physical properties alone, while serving to distinguish between many anthracites and semianthracites, fail when applied to borderline coals.

Friability and Specific Gravity

All friability tests on these coals were made by H. F. Yancey, U. S. Bureau of Mines, Northwest Experiment Station. His data are used with the permission of the U. S. Bureau of Mines in the construction of Fig. 1, which shows the lack of relation of volatile matter to friability. The fact that the semianthracites are more friable than most anthracites is illustrated by Fig. 1, but the close proximity of some of the anthracite and semianthracite points makes it impossible to use friability, thus determined, to define border-line cases. The writer feels that a more significant relationship would be shown if friability tests were made on one definite size of coal (nut coal) as it comes from the breaker.

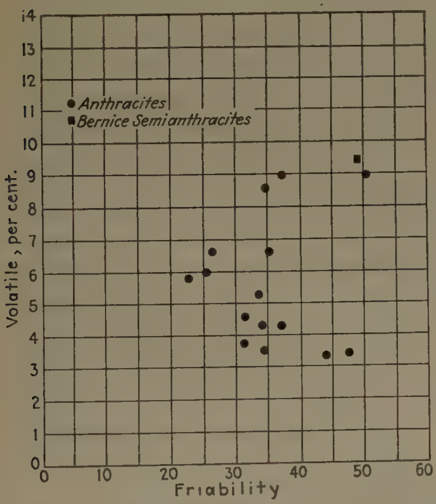

Fig. 1.—Volatile content (moisture and ash free) versus friability.

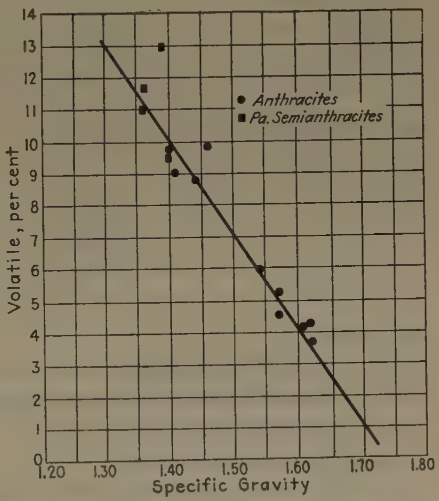

Fig. 2.—Volatile content (moisture and ash free) versus (corrected) specific gravity.

The specific gravities that are of the greatest practical value to the producers and consumers of anthracite are the apparent gravities rather than the true gravities. For this reason, and for the further reason that the determination of true specific gravity is very difficult in the average laboratory, a modified method was chosen as follows:

Three hundred grams of coal, previously crushed to pass ¼-in. screen, were air-dried and weighed to ±0.1 gram. This coal was then placed in a weighed and calibrated Pyrex liter flask marked so as to show a capacity of 1050.0 grams of boiled distilled water at 25° C.

The flask was next half filled with distilled water and boiled for 10 min. to remove air. It was then cooled to 25° C. in a water bath, filled to the mark on the neck with previously boiled and cooled distilled water and weighed. The specific gravity was calculated by the use of the following formula:

$$\text{Sp. gr.} = \frac{W}{W - (W_1 - P)}$$

in which

W = weight of coal in grams (air dry),
W_1 = weight of coal, bottle and water to fill,
P = weight of bottle and water to fill.

The figures obtained by this method can be checked to the second decimal place.

When the specific gravities thus secured were plotted on the isovol map it was evident that there exists a close relationship between specific gravity and volatile matter. To iron out some of the irregularities, the specific gravities were calculated to a moisture-and-ash-free basis using the formulas:

$$\frac{100}{g} = \frac{M}{1.00} + \frac{100 - M}{gd}$$

$$\frac{100}{gd} = \frac{Ad}{2.7} + \frac{100 - Ad}{gc}$$

in which

M = air-dried moisture, per cent
g = specific gravity as determined
gd = specific gravity dry
Ad = dry ash, per cent
gc = specific gravity, moisture and ash free

These specific gravities are shown in Table 3, where the analyses are arranged in the order of increasing volatile, reading from top to bottom. The relationship is shown in the form of a plot in Fig. 2. There is a good relationship in this instance. All the anthracites are 1.40 and higher (moisture-and-ash-free basis), and all the semianthracites are 1.40 and lower. But alas, one anthracite and one semianthracite fall on the same line. This anthracite has 91.6 per cent F.C. on the dry 1.1 ash-free basis as compared to 91.5 for the semianthracite in question. This demonstrates the fact, however, that the figure 92.0 F.C. as proposed in the classification is a logical boundary figure.

CLOSED-TUBE TEST

In mineralogy, substances are often tested by heating in a glass tube closed at one end, and identified by observing the nature of the sublimate that condenses on the upper cool walls of the tube. This test was tried on the semianthracites and high-volatile anthracites. When modified as indicated below it was found that all semianthracites gave a tar deposit while the high-volatile anthracites failed to give a deposit. One high-volatile anthracite, however, just failed to give a deposit. "Just failing to give a deposit" means that it gave a similar bituminous odor, suggesting a possible trace of tar. This particular sample is the one listed in Table 5 as "Shamokin district, maximum volatile." It has a fixed carbon on the dry 1.1 ash-free basis of 91.6 as compared to 91.5 for the Bernice semianthracite, which gave a tar deposit. The reason for its failure to yield tar is probably to be found in the fact that it is lower in hydrogen than the Bernice coal (Table 4).

TABLE 3.—*Average Proximate Analyses and Specific Gravities of Face Samples, Anthracite Region*

(Moisture-and-ash-free)

Field	Number of Samples	V.M.	F.C.	Air-dry Sp.Gr.	Corrected Sp. Gr. (M/A)
Eastern Middle	24	3.7	96.3	1.65	1.62
Southern (Panther Creek district)	13	4.2	95.8	1.65	1.61
Southern (East Schuylkill district)	6	4.3	95.7	1.68	1.62
Western Middle (West Mahanoy district)	21	4.6	95.4	1.64	1.57
Southern (West Schuylkill district)	10	5.3	94.7	1.64	1.57
Northern	100	6.1	93.9	1.56	1.54
Southern Fishtail[a] (Gold Mine Run)	1	7.3	92.7	1.45	1.41
Southern (Lykens district)	14	8.8	91.2	1.52	1.44
Western Middle (Shamokin district)	16	9.0	91.0	1.49	1.41
Shamokin district,[a] max. vol.	1	9.8	90.2	1.49	1.40
Lykens district,[a] max. vol	1	9.9	90.1	1.55	1.46
Average		5.8	94.3	1.60	1.55
ANTHRACITE REGION—SEMIANTHRACITE					
Western middle field (Trevorton)	2	11.7	88.3	1.43	1.36
SULLIVAN COUNTY COALS—SEMIANTHRACITE					
Bernice (Connell mines)	3	9.5	90.5	1.46	1.40
Laporte (Rock Run mine)	1	11.0	89.0	1.43	1.36
Forksville (Monahan mine)	1	13.0	87.0	1.55	1.39

[a] Not included in average.

The closed-tube test is as follows: Six grams of coal, crushed to pass 60 mesh, are placed in the closed end of a Pyrex tube 7 in. long, 5/8 in. in diameter, and with walls 1/16 in. thick. A plug of glass wool about one inch thick is pushed down the tube to within 1/2 in. of the coal. The closed end of the tube is held in a 6-in. Bunsen flame (530 B.t.u. gas) about one inch above the green cone of unburned gas and heated for seven minutes. The maximum temperature of the coal in the tube, as indicated by a chromel-alumel thermocouple, was about 675° C. The tube is rotated and tapped with the fingers during the heating. During the last minute of heating the flame is directed to the zone just below the wool plug. If tar is present, it will condense as a brown yellow sublimate in the glass wool or just above it. This color will not fade when the tube cools. High-sulfur coals will give a similar color, which will fade to a white when cold.

This test gave positive results on all the semianthracites listed in Table 4 and in addition on six semianthracites from Virginia.

TABLE 4.—*Average of Ultimate Analyses of All Face Samples, Anthracite Region*

(Moisture-and-ash-free)

Field	*H*	*C*	*N*	*O*	*S*	B.t.u.
Eastern Middle	2.1	94.7	0.8	1.6	0.8	14,828
Southern (Panther Creek district)	2.1	95.0	0.8	1.4	0.7	14,823
Southern (East Schuylkill district)	2.0	94.8	0.8	1.7	0.7	14,802
Western Middle (West Mahanoy district)	2.3	94.3	0.9	1.5	1.0	14,867
Southern (West Schuylkill district)	2.0	94.8	0.8	1.7	0.7	14,898
Northern	2.8	92.4	1.1	2.5	1.2	15,038
Southern Fishtail[a] (Gold Mine Run)	3.4	93.6	1.1	1.1	0.8	15,580
Southern (Lykens district)	3.3	92.0	1.3	2.6	0.8	15,237
Western Middle (Shamokin district)	3.6	91.2	1.5	2.6	1.1	15,258
Shamokin district,[a] max. vol	3.7	91.3	1.5	2.6	0.9	15,270
Lykens district,[a] max. vol	3.1	92.5	1.1	2.7	0.6	15,100
Average	2.5	93.7	1.0	2.0	0.9	14,968
ANTHRACITE REGION—SEMIANTHRACITE						
Western Middle (Trevorton)	4.1	90.8	1.6	2.5	1.0	15,440
SULLIVAN COUNTY COALS—SEMIANTHRACITE						
Bernice (Connell)	3.8	92.0	1.2	2.2	0.7	15,463
Laporte (Rock Run)	4.0	90.9	1.1	3.3	0.7	15,460
Forksville (Monahan)	4.2	88.9	1.1	5.1	0.7	15,160

[a] Not included in average.

The Forksville coal just began to agglutinate in the tube, which suggests that the lower limit for semianthracites is probably close to the figure 86 as proposed in Table 1. Further tests might show the fitness of this figure by observing the point at which the powdered coal in the tube formed a coherent mass.

This test again justifies the proposed figure of 92.0 per cent F.C., for with two exceptions it shows tar from all coals below 92.0 and no tar from all coals 92.0 or above. The two exceptions below 92.0 were individual anthracite samples of 91.6 F.C., which did not yield tar. These two samples, however, were from groups of samples that averaged above 92.0, so from the standpoint of averages 92.0 is exactly where the boundary falls in classifying all the coal as shipped from the mines.

For the classification of individual beds or separate benches of beds, however, there will no doubt be a few samples ranging between 91.5 and 92 F.C. on the dry 1.1 ash-free basis where some such test as the closed-tube test will have to be made to decide whether the coals in question shall be classed as semianthracite or anthracite. In these cases, the author would suggest that the coals that fail to yield tar in the closed tube be classed as anthracite even though they fall somewhat below the figure 92.

COMMENTS ON ANALYSES

The anthracite analyses listed in this paper, with exceptions as indicated by number of samples in Table 3, are weighted averages from many fresh samples. The semianthracite analyses are all from single face samples with the exception of Bernice, where three face samples were obtained. Failure to obtain more semianthracite was due to the meagerness of mine development in the region where this coal occurred.

Many of the comments made here refer to the original 428 analyses, which are too numerous to record in this short paper.

The ultimate analyses given in Table 4 show a consistent increase in hydrogen as one goes from low-volatile anthracite to semianthracite. The oxygen content increases also, but the increase is not consistent.

The analyses in Table 5 show that all the analyses for anthracite with the exception of the two individual face samples fall within the limits of 92.0 F.C. plus, minus 98 F.C. as proposed in Table 1. The maximum difference in volatile matter on a moisture-and-ash-free basis of beds in any one colliery was found to be 3.5 per cent.

The average of the maximum differences in volatile matter on a moisture-and-ash-free basis for all collieries was found to be 2.2 per cent.

The average of the maximum deviations from the average volatile matter on a moisture-and-ash-free basis for all collieries was found to be ±1.1 per cent.

The latter figure is a reasonable tolerance for the adjustment of isovol points where only one face sample could be obtained. It was used in one case only and that was in connection with the one face sample obtained at Gold Mine Run in the Southern Fishtail. The 7.3 figure given in the analysis did not fit in well with the other points. It is reasonable to assume that this figure would be about 8 had several face samples been obtained, especially samples of other beds at this location. In passing, it is of interest to note that this coal is anthracite

TABLE 5.—*Average Proximate Analyses of All Face Samples, Anthracite Region*

	H_2O	V.M.	F.C.	Ash	F.C. 1.1 × Ash Free Basis	Field
As received.......	5.3	3.2	83.1	8.4	91.6	Eastern Middle
Dry..............		3.4	87.7	8.9	97.2	Eastern Middle
As received.......	3.5	3.7	81.0	11.8	93.1	Southern
Dry..............		3.8	84.0	12.2	97.0	East Schuylkill district
As received.......	3.5	3.7	83.2	9.6	93.0	Southern field
Dry..............		3.8	86.2	10.0	96.8	Panther Creek district
As received.......	3.7	3.9	79.9	12.5	92.6	Western Middle
Dry..............		4.0	83.0	13.0	96.8	West Mahanoy district
As received.......	5.2	4.4	79.0	11.4	90.3	Southern
Dry..............		4.7	83.3	12.0	96.0	West Schuylkill district
As received.......	5.0	5.2	80.5	9.3	89.7	Northern
Dry..............		5.5	84.7	9.8	95.0	Northern
As received[a].......	2.2	6.6	84.2	7.0	90.9	Southern Fishtail
Dry..............		6.8	86.1	7.1	93.4	Gold Mine Run
As received.......	3.4	7.4	77.0	12.2	88.9	Southern
Dry..............		7.7	79.7	12.6	92.6	Lykens district
As received[a].......	2.6	8.3	76.2	12.9	88.8	Lykens district
Dry..............		8.5	78.2	13.3	91.6	Maximum volatile
As received.......	2.3	7.8	78.8	11.1	89.8	Western Middle
Dry..............		8.0	80.6	11.4	92.2	Shamokin district
As received[a].......	2.5	8.3	76.3	12.9	89.3	Shamokin district
Dry..............		8.5	78.2	13.3	91.6	Maximum volatile
Average						
As received.......	4.0	4.9	80.3	10.8	91.1	
Dry..............		5.1	83.7	11.2	95.5	
ANTHRACITE REGION—SEMIANTHRACITE						
As received.......	1.7	10.1	76.7	11.5	87.4	Western Middle
Dry..............		10.3	77.9	11.8	89.7	Trevorton
SULLIVAN COUNTY COALS						
As received.......	2.8	8.3	79.6	9.3	88.6	Bernice
Dry..............		8.5	81.9	9.6	91.5	Connell Mines
As received.......	2.0	9.6	77.8	10.6	88.1	Laporte, Rock Run mine
Dry..............		9.8	79.4	10.8	90.0	Laporte, Rock Run mine
As received.......	1.2	10.1	67.7	21.0	88.0	Forksville, Monahan mine
Dry..............		10.2	68.6	21.2	89.4	Forksville, Monahan mine

[a] Not included in average.

rather than a semianthracite, for the statement has often been made that this Southern Fishtail coal is semianthracite.

The maximum difference in volatile matter on a moisture-and-ash-free basis of different face samples from the same bed in the same colliery was found to be 2.3 per cent.

The average difference in volatile matter on a moisture-and-ash-free basis of different face samples from the same beds in the same colliery was found to be 0.6 per cent for the whole region.

HIGH-VOLATILE AND LOW-VOLATILE ANTHRACITES

At these meetings in 1930, Marius R. Campbell[2] proposed a classification which divided the anthracitic group into three classes as follows: class A, or high-volatile anthracite; class B, or low-volatile anthracite; and class C, or meta-anthracite. He would eliminate the term "semianthracite," partly because he feels that the term is misleading and partly because he feels that high-volatile anthracites and semianthracites are so nearly alike that there is no good ground for separating them.

The writer has shown that there are significant differences between high-volatile anthracites and semianthracites and that these differences justify the name "anthracite" and "semianthracite." The writer feels that the prefix "semi" is a proper one, for it is defined in Webster's New International Dictionary as meaning "akin to, half, partly, and imperfectly." If we accept the interpretation "akin to," the name semianthracite should meet with no objection.

The most serious objection to this name in the past has been its conflict with "semibituminous," a term used to designate a coal higher in rank than bituminous, whereas semianthracite designated a coal lower in rank than anthracite. When both of these terms are used in this way, the writer strongly objects to them. He can find no fault with the term semianthracite, if semibituminous is to be dropped.

GEOLOGICAL SIGNIFICANCE

The data set forth in this paper are in such close agreement with David White's[3] theory of progressive regional carbonization that the author would like to call attention to a few significant facts as a tribute to David White.

The fact that the isovols shown in the accompanying map (Fig. 3) cross the present anticlinal and synclinal axes in all fields except the northern one indicates that carbonization was brought about by horizontal thrust pressures applied prior to and in a different direction from the immediate forces that caused the present anticlines and synclines.

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[2] M. R. Campbell: Natural Groups of Coal and Allied Fuels. *Trans.* A. I. M. E. (1930) **88**, Coal Division.

[3] D. White: Progressive Regional Carbonization of Coals. *Trans.* A. I. M. E. (1925) **71**, 253.

The discovery of high-volatile semianthracite at the Steinhart colliery near Trevorton in steeply pitching coal beds, which are highly twisted and in parts quite schistose, shows that the immediate forces

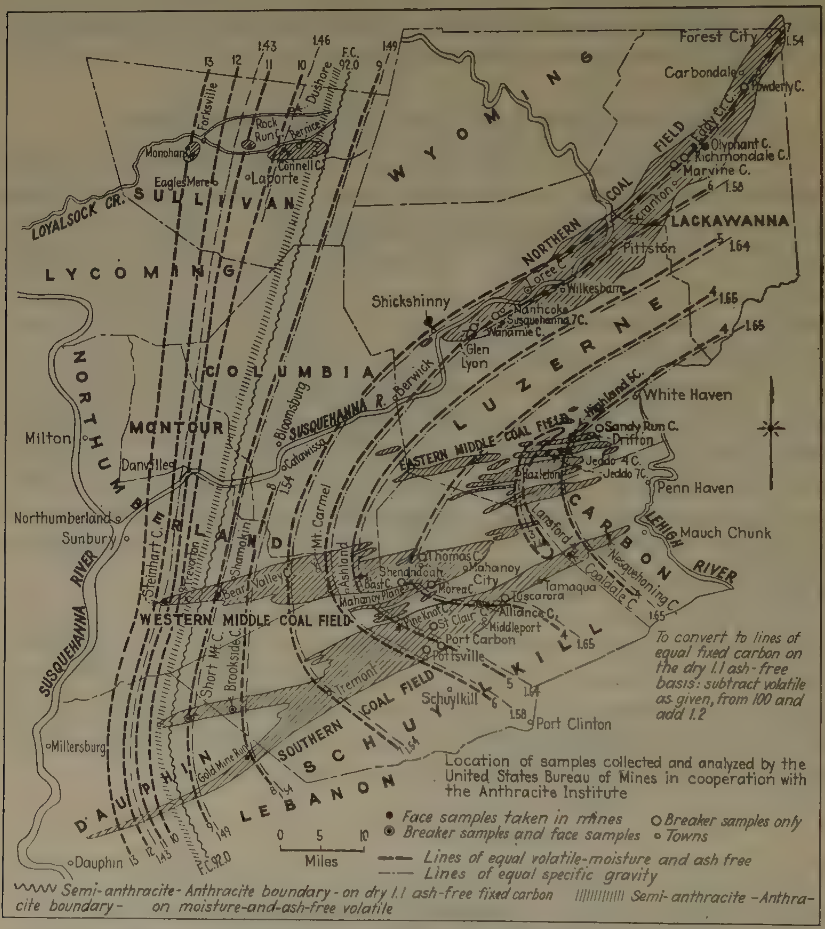

FIG. 3.—LOCATION OF SAMPLES COLLECTED AND ANALYZED BY THE U. S. BUREAU OF MINES IN COOPERATION WITH THE ANTHRACITE INSTITUTE.

that caused the folds did not alter the coal chemically to any great extent. This is further substantiated by the occurrence of the physical counterpart of this coal in areas of low-volatile and medium-volatile matter.

This steeply pitching Steinhart colliery coal at the western end of the Western Middle field is almost identical in composition with the flat-lying semianthracites west of Bernice.

The lack of relationship of friability to volatile matter suggests that the immediate forces that caused the present folds had little to do with carbonization except in a negative way. These folds no doubt relieved the horizontal pressures so as to prevent the metamorphism from progressing to the formation of graphite.

Last and weakest is the indication that the law of Hilt still holds in the anthracite fields. In 9 out of 11 collieries selected from the five major fields, it was found that the volatile content decreased with original depth of beds in spite of their present position.

From the foregoing it seems reasonable to conclude that anthracitization was practically, if not entirely, complete before the existing folds were formed.

Conclusions

The figures arrived at by the Coal Classification Committee as marking the boundary lines between the classes of the anthracitic group were largely derived by plotting all analyses of these classes that could be found in the literature. Many of these analyses were of doubtful value because of lack of history of the samples. In spite of these doubts, the figures arrived at seem to be correct.

The present paper presents the averages of over 200 face samples taken in the standard way and analyzed by standard methods. These analyses are consistent with the figures already arrived at.

In addition to supporting the proposed figures through the presentation of many new reliable analyses, further support of these figures is given by showing that the coals thus classified actually differ in physical properties and tar yields.

The map on which are plotted lines of equal volatile matter and lines of equal specific gravity shows that the proposed classification has the merit of separating the semianthracite and anthracite fields into two geographically distinct units.

The boundary line between semianthracite and anthracite has the further merit of grouping these coals into classes that are consistent with past and present commercial classifications.

The final classification of the anthracitic group as arrived at by this study would be as shown in Table 6.

Probably 99 per cent of all the anthracitic coals of North America can be placed definitely by this classification, but there are bound to be a few that will overlap so that additional factors will have to be used in placing them in one class or another. In such cases I believe that it would be better to use additional properties rather than tolerances in the figures set for fixed carbon. In the few cases of border-line coals, for instance, which fall between 91.5 and 92.0 F.C., the closed-tube test described herein should be used as a deciding test. Since these cases will be relatively rare, it would seem best to provide for them in this way rather than to alter the main classification, which, as it stands, will apply in practically all cases.

TABLE 6.—*Proposed Classification of Anthracitic Group*

Group	Class	Limits of Fixed Carbon,[a] Mineral-matter-free	Requisite Physical Properties
Anthracitic	Meta-anthracite	Dry F.C., 98+	Nonagglutinating
	Anthracite	Dry F.C., 92+, −98	
	Semianthracite	Dry F.C., 86+, −92	

[a] Coals between 91.5 and 92.0 F.C. on the above basis are anthracites if they do not yield tar in the standard closed-tube test.

ACKNOWLEDGMENTS

The writer wishes to acknowledge the cooperation of the School of Mineral Industries, The Pennsylvania State College, State College, Pa., and the valuable assistance of A. C. Fieldner, C. M. Stull, H. A. Goodman, and H. M. Cooper of the United States Bureau of Mines.

DISCUSSION

(*T. W. Harris, Jr. presiding*)

E. L. BULLER,* Scranton, Pa.—I should like to have a definition of friability. There would seem to be a definite relation between specific gravity and hardness to volatile content, and a definite relation between specific gravity and denseness to friability, if friability is taken to mean breakage or degradation. Brittleness increases as the apparent specific gravity increases.

H. G. TURNER.—There were too many inter-relations involved to give a specific answer. I used the friability figures obtained by Yancey, who employed a tumbling jar equipped with shelves, using lump coal which had been carefully packed and shipped. Hardness differs from toughness; a soft coal may be tough and hence less friable than a harder coal.

T. A. HENDRICKS,† Washington, D. C.—The friability in semianthracite seems to be dependent on the dissemination of the ash content. As to the isovol map, two opposing interpretations could be put on the same data, and a series of analyses across one of the basins, normal to its axis is needed. I should expect to find a low volatile content in the center with higher volatile content on each flank. I agree that Hilt's law should hold, although its effects are obscure in bituminous coal, owing to variation in types of coal.

H. G. TURNER.—With respect to Hilt's law, anthracitization has erased differences in original composition. As to friability and ash, a bone coal containing 30 per cent ash was found to be tougher and less friable than a purer coal.

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* Combustion Engineer, Hudson Coal Co.

† U. S. Geological Survey.

Probably no two persons would draw exactly the same isovol map using the same data. The lines on my map were generalized in very much the same way that the weather man generalizes isobars and isotherms. The resulting lines are consistent with existing isovol maps of the other coal regions of the state. One fact that leads me to feel that my map is fairly accurate is that analyses obtained since the map was made fall in their proper place when plotted on the map.

E. STANSFIELD,* Edmonton, Alta.—The method for the determination of apparent specific gravity appears open to question. The coal is air-dried, which presumably will cause minute fissures filled with air, and later boiled for 10 min. to remove air. There is, however, no assurance that some air or gas originally in the coal is not also removed and replaced by water. Would it not be better to omit both air-drying and boiling? In the calculation of the apparent gravity of the dry coal the assumption is made that each gram of water present has increased the size of the lump by 1 cu. cm. We know that many coals shrink when they dry, but has anyone yet determined the apparent volume of water in coal? This point is of interest, although relatively unimportant with low-moisture coals such as anthracite.

H. G. TURNER.—With coals other than anthracite, more precautions would need to be observed in the determination of specific gravity. Anthracite does not become fissured when air-dried.

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* Chief Chemical Engineer, Research Council of Alberta.

Relative Efficiencies of Domestic Fuels in Relation to Their Costs and Their Advantages

By Ralph A. Sherman,* Columbus, Ohio

(Columbus Meeting, October, 1933)

In these days of intense competition for the domestic fuel market much is heard of the low efficiencies with which solid fuels, and most particularly bituminous coal, are burned in domestic heating equipment. In contrast to the every-day operating efficiencies of 85 and 90 per cent in large, central, electric-generating stations, efficiencies of 50 per cent and lower are cited for the hand-fired home furnace.

Just as the miles per gallon of gasoline is only one of the factors that determine the cost of operation of an automobile, so the thermal efficiency is only one factor in the cost of domestic or other heating. The purpose of this paper is to analyze the total costs of domestic heating by various fuels and types of equipment, to determine the relative importance of efficiency and other factors, and to discuss what this signifies relative to the future of the use of fuels in the home.

FACTORS THAT GOVERN THE CHOICE OF FUEL AND BURNING EQUIPMENT

A rational, intelligent selection of fuel and equipment for heating the home would consider two groups of factors:

Cost Factors

1. Cost of potential heat in fuel
2. Efficiency of combustion and heat transfer
3. Fixed charges
    - *a.* Interest
    - *b.* Depreciation
    - *c.* Repairs
4. Power costs
5. Cost of handling ash and refuse.

Convenience Factors

1. Attention
2. Dirt
3. Noise

As just stated, the purpose of this paper is to analyze the relative importance of the five cost factors and not to discuss particularly the relative total costs of different methods of heating. However, to arrive at the percentage costs and to show clearly how they were determined, a statement of the total costs is unavoidable. One example only will be given, that of Columbus, Ohio, where the writer is familiar with conditions. Columbus lies near various coal fields, has a favorably low rate for natural gas, and fuel oil is available although not at an exceptionally low cost. Conditions in Columbus cannot, therefore, be said to be typical, and quite different relations may be found for other cities. It would burden the paper with tables to attempt to cover a really typical group; the one example, and it must be considered only as an example, will show clearly the principles and the necessary modifications for other conditions. It is hoped that discussers of the paper will present such analyses for other districts.

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* Fuel Engineer, Battelle Memorial Institute.

Assume the case of a homeowner in Columbus who has used, on an average, 15 tons of Pocahontas lump coal, hand-fired, in the steam-heating system of his eight or nine-room house for the past several years. He now is considering whether he should change to a stoker, an oil burner, or a conversion burner for natural gas. The total cost of heating his home will be analyzed in four ways.

TYPES OF FUELS AND THEIR COSTS

A wide variety of fuels is available in Columbus, but, as a previous analysis(1)* showed that there was a spread of only about 8 per cent in the fuel cost, based on useful heat, among all the bituminous coals commonly on this market, the argument may be further simplified by a consideration of but two coals, Pocahontas lump and Hocking Valley stoker coal, and of oil and natural gas. Table 1 gives the characteristics of these fuels. At the time of preparation of this paper, Sept. 1, 1933, coal prices were in an unstable condition because the operators' and retailers' coal codes were not yet finally written. Some increase was expected, so the Pocahontas coal was set at $7.25 and Hocking at $4.50 per ton, although the prices then current were about $6.75 and $4.25. At the time of presentation of this paper the prices may be somewhat different, but the principles discussed will remain the same.

The fact that Pocahontas, low-volatile coal was selected as an example for hand-firing and Hocking Valley, high-volatile coal for stoker firing is not to be considered as a specific recommendation for this or any district. It was regarded as logical that a man who would consider an investment in automatic heating would previously have used what he considered a premium fuel, a so-called "smokeless" coal. For the stoker example a low-volatile coal could also have been used, as both high and low-volatile coals are used successfully on underfeed stokers; at prevailing Columbus prices the costs would probably not be greatly different.

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* Superior figures in parentheses refer to references at end of paper.

TABLE 1.—*Characteristics of Fuels, as Fired*

Fuel	Ash, Per Cent	Calorific Value, B.t.u.	Cost	B.t.u. per Dollar
Pocahontas lump...	4.6	14,600 per lb.	$7.25 per ton	4,030,000
Hocking stoker.....	7.0	12,400 per lb.	4.50 per ton	5,510,000
Natural gas........		1,000 per cu. ft.	0.55 per 1000 cu. ft.	1,820,000
No. 3 fuel oil.......		141,000 per gal.	0.065 per gal.	2,170,000

Efficiency of Combustion and Heat Transfer.—Table 2 shows the efficiency of the boiler that probably will be obtained with the four fuels.

TABLE 2.—*Thermal Efficiencies Assumed*

Fuel	Method of Burning	Efficiency, Per Cent
Pocahontas lump.........................	Hand-fired	50
Hocking stoker..........................	Underfeed stoker	60
Natural gas.............................	Conversion burner	70
No. 3 fuel oil............................	Oil burner	65

The efficiencies given are the percentages of heat in the fuel delivered in the steam at the outlet of the boiler. Some people consider that the efficiency of a domestic heating unit should be measured as the entire heat delivered to the house, calculated as the difference between the heat in the fuel and the loss in the unburned fuel to the ash pit and the heat in the gases at the top of the chimney. The research of the National Warm Air Heating and Ventilating Association at the University of Illinois has shown(2) that, measured in this way, the efficiency with anthracite averaged 95 per cent and with bituminous coal 75 per cent. The writer is not convinced that this is a fair method of computation because the function of any heating system is to put the heat when and where it is required. The knowledge that the over-all efficiency of the furnace and house is 80 per cent is small comfort if the basement is at 80° and the northwest bedroom is at 60°.

Some disagreement with the assumed efficiencies is to be expected, but they have been taken as representative of what can and should be obtained in actual use conditions, where they are generally lower than in laboratory tests. The Bureau of Mines(3) has shown that the efficiency of domestic boilers hand-fired with semibituminous coal averaged 56 per cent; to be conservative, the value of 50 per cent has been assumed. The efficiencies assumed for the stoker, oil burner and conversion gas burner are based on various published data(4,5,6) with such modifications as dictated by private information from reliable sources and the writer's judgment so as again to be conservative.

A domestic underfeed stoker will give 65 or 70 per cent efficiency under favorable conditions, but 60 per cent is probably a liberal estimate for the intermittent operation and probable lack of attention it will receive in the average small house. Domestic oil burners can give efficiencies of 70 per cent and over, but without careful adjustment as to furnace draft the excess air may be so high that efficiencies lower than the 65 per cent assumed are obtained.

Fixed Charges on Equipment.—The annual fixed charges on the combustion equipment, which consist of the interest on the investment and a fraction of the investment based on the years of expected life, are rarely considered by a homeowner as a part of his heating cost. This is true because little of the equipment of a home is bought with a full consideration of the total cost and the anticipated return. The fixed charges on the entire heating system, boiler and radiators, might also be included, but as they are the same for all the fuels considered they have been omitted from this analysis.

The fixed charges are set at 15 per cent annually for all equipment. This is made up of a 10 per cent depreciation charge, 2 per cent for repairs, and 3 per cent interest. The 3 per cent interest charge is the approximate mean over the 10-year period when 6 per cent is charged on the undepreciated balance. Domestic stokers and burners have not been in use long enough to fix an accurate average life; therefore, they have been depreciated over 10 years, as is the normal practice for industrial equipment.

Power Consumption.—The electric current used by a stoker or oil burner is usually recorded on the same meter as light and other household power requirements and is not normally added to heating costs. The average values used are 15 kw-hr. per ton of coal for the stoker(7) and 7 kw-hr. per 100 gal. of oil for the oil burner;(8) power is calculated to cost 5¢ per kilowatt-hour.

Ash Handling.—The cost of the removal of ashes is another small cost that is generally lost in other petty household expenses. The cost is subject to considerable variation but has been calculated as 10¢ per bushel of 40 lb., an average cost in Columbus.

Comparison of Total Heating Costs.—Table 3 shows the summary of the heating costs and the percentage of the total represented by the various items for the assumed home that burned 15 tons of Pocahontas coal when hand-fired. The amounts of the other fuels necessary to deliver the same heat requirements were calculated from their efficiency relative to hand-fired coal.

In addition to these costs, which can be set down definitely in dollars and cents, there are numerous intangibles, which have definite value but which cannot be averaged and therefore have been classed under the convenience factors. They are the dirt, which includes cost of cleaning walls, drapes, rugs, and laundry of linens; the value of added space in the basement when no coal bin is required, as with gas and oil; and the value from a health standpoint of uniform, thermostatically controlled temperature, which is obtained with stoker, oil, or gas but which cannot successfully be applied to hand-fired coal.

TABLE 3.—*Summary of Heating Costs, Columbus, Ohio, House*

Fuel	Pocahontas		Hocking		No. 3 fuel oil		Natural gas	
Method of firing	Hand		Stoker		Oil burner		Conversion burner	
Thermal efficiency, per cent	50		60		65		70	
Fuel used per year	15 tons		14.7 tons		2390 gal.		313,000 cu. ft.	
Cost of fuel	$7.25		$4.50		6.5c per gal.		55c per 1000 cu. ft.	
Cost of equipment			$340		$350		$145	
Costs:	Dollars	Per Cent	Dollars	Per Cent	Dollars	Per Cent	Dollars	Per Cent
Fuel	$108.75	96.8	$ 66.15	49.6	$155.35	71.8	$185.35	89.5
Power			11.00	8.3	8.37	3.9		
Ash removal	3.45	3.2	5.10	3.8				
Total fuel costs	112.20	100.0	82.25	61.7	163.72	75.7	185.35	89.5
Fixed charges, 15 per cent			51.00	38.3	52.50	24.3	21.75	10.5
Total yearly cost	$112.20	100.0	$133.25	100.0	$216.22	100.0	$207.10	100.0

Relative Importance of Efficiency.—In the four examples of Table 3, the fuel cost is 62 per cent for stoker firing to 100 per cent for hand firing, of the total cost of heating. The smaller the percentage of the total cost that is represented by the fuel cost the less is the importance of efficiency. Thus, an increase of 10 per cent in the efficiency of combustion will result in a decrease of 10 per cent in the cost of heating when firing by hand, but only about 6 per cent with the stoker. With oil and gas the thermal efficiency is increasingly important. However, in Columbus the cost of heat in the form of gas and oil is so high relative to coal that, if the efficiency were 100 per cent, their costs would be higher than that of coal.

COAL A LOW-COST FUEL

This tabulation shows that coal is the low-cost method of domestic heating in the Columbus district. Based on the comparatively low cost of natural gas in Columbus, Pocahontas coal for hand firing would have to sell at $13.60 and Hocking stoker coal at $9.50 per ton to have a heating cost equal to that of gas. Or, natural gas would have to sell at 29¢ per 1000 cu. ft. to have a heating cost equal to that of Pocahontas hand-fired or at 36¢ to equal that of Hocking stoker coal at the coal prices given in the table.

On the Pacific Coast, and in the Southwest and Northwest, and possibly in New England and some parts of the Atlantic seaboard, where freight rates on coal are high, gas or oil will be found to cost less than coal. But in many populous districts coal will be found to show a lower heating cost than oil and gas. Quite obviously, therefore, thousands of domestic consumers of coal who have turned to gas and oil have not changed because of lower cost of heating. Nor have they changed because of the possible higher thermal efficiencies. Few people are so much interested in the conservation of these natural resources, but they *are* interested in the conservation of their time and effort and an evenly warm, clean home.

For the market represented by this type of homeowner, therefore, the bituminous coal industry should direct its efforts not so much toward the development of equipment that will give a higher thermal efficiency as toward the development of equipment that will give the qualities, convenience and cleanliness that the public now looks for in oil and gas.

Domestic stokers now on the market go a long way toward meeting these requirements as contrasted to the hand-fired furnace, but further refinements are needed. Coal still must be shoveled into the hopper and ashes and clinker must be removed. Stokers are not equally suited to all coals and a knowledge of the suitability of various coals is lacking.

Although they do not completely emancipate the householder from the shovel and poker unless he pays for the service (a complete heat service is now available in many cities for $5 or so a month), stokers for bituminous coal do possess a feature, more important now than some five years ago, that of economy. For our particular example, although there was a saving of $30 in fuel cost over hand firing, the fixed charges more than overcome the saving. If, however, the stoker were to last longer or need fewer repairs, if the house were enough larger or the heating season longer, or if the differential in cost of lump and stoker coal were greater, actual savings on total heating costs could occur. In larger installations, as apartments, hotels, laundries, et cetera, where the stoker is used over longer periods and with more continuous operation, so that the thermal efficiency may be 70 per cent or more and the fixed charges a smaller fraction, large money savings are made and the equipment often pays for itself in much less than the 10 years we have here assumed to be its life.

A point worthy of attention is that the stoker is the only one of the three types of mechanical equipment that has a chance to pay for itself. If our assumed Columbus homeowner buys an oil or gas burner, he not only pays for that equipment but must pay annually for fuel alone more than he would pay for coal hand-fired.

The need is evident for stokers or other automatic combustion equipment for coal at a lower first cost, so that the fixed charges may be reduced. The low first cost of the conversion equipment for natural gas is an obvious inducement to change to that fuel.

Heating equipment is used for but 6 to 7 months of the year. The cooking field has been almost entirely relinquished to gas. We have seen the all-electric home, the all-gas home, and the anthracite industry has followed with an all-anthracite home that is well described in the booklet, "Tomorrow's House," published by the Philadelphia and Reading Coal and Iron Co. Although greater difficulties are obvious, the possibilities exist of the year-round use of bituminous coal in the home. It would be used, as is anthracite, for cooking in an improved, well insulated range, and in a steam boiler that supplies: (1) steam to an air heater for winter heating, (2) steam for winter humidification, (3) steam for the energy required for an absorption-type domestic refrigerator, (4) steam for an absorption or steam-ejector system for dehumidification and cooling of air in the summer, and (5) hot water for the entire year.

The market for this type of equipment is a small percentage of the total coal market, but it affords interesting possibilities in greatly increasing the use of coal in that market and cannot be neglected.

Summary

The particular points that it has been desired to emphasize in this analysis of heating costs are:

1. That fixed charges are an important, though often neglected, fraction of the total cost of domestic heating with the use of any mechanical equipment.
2. That the greater the percentage of the total cost that is represented by the fixed charges, the less important is the thermal efficiency of the system.
3. That coal, hand-fired or stoker-fired, is, in a large part of the United States, cheaper than gas and oil, and that these fuels have been and are being adopted not because of higher thermal efficiency but because of anticipated advantages of convenience and cleanliness.
4. That developments in automatic coal-burning equipment must be directed toward the attainment of these advantages.
5. That the present desirability of coal because of low operating cost can be enhanced by the development of equipment of lower first cost.
6. That the use of coal in a certain small market may be increased by the development of equipment for the year-round use of coal in the home.

In closing, may I depart somewhat from the subject to call attention to the fact that the majority of the domestic users of coal are not likely prospects for automatic coal-burning equipment; they are tenants and homeowners of the economic class that cannot afford to buy equipment and must use the cheapest available fuels. It is this large class of domestic users that makes and will continue to make our communities dirty with smoke even after clean automatic equipment is developed for those who can afford to put it in. I do not venture here to suggest the method, but I do want to point to the need for low-priced auxiliary equipment or a low-cost method of processing coal that will prevent that smoke and take from bituminous coal its bad name as the source of our cities' dirt.

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DISCUSSION

(*Charles E. Lawall presiding*)

G. R. DELAMATER,[*] Cleveland, Ohio.—I notice that the figures given for natural gas are only for conversion burners, which I understand to be a burner that could be installed in a furnace that was designed and built for the burning of solid fuels. In order to get the entire picture, we should have the figures of the cost of gas, and the efficiency of the burning of gas in furnaces built especially for gas, which I believe are much higher in efficiency.

In Cleveland a great many homes are using special gas furnaces, and many that have furnaces built for solid fuels are using conversion burners such as the Barber burner. I have no definite data, but I have noted on several occasions that the apparent increased cost of burning gas in a furnace built for solid fuel is at least one-third more—perhaps even greater—than burning gas in a furnace designed for gas.

This seems to be almost a direct comparison with the anthracite burner that operates at very high efficiency but in which the furnace used is designed especially for this anthracite coal. If Mr. Sherman has anything on this comparison, I should be interested to hear it.

C. SCHOLZ,[†] Charleston, W. Va.—I think I can answer Mr. Delamater's question. I moved into a new house in Charleston, equipped with a combination furnace, to use either gas or coal; my next door neighbor had a duplicate of my house. After the first winter, I concluded that my heating cost was too high and I installed a gas furnace. Three years later I made a check and found that my bills for that period were about 10 per cent below those of my neighbor, who continued to use the combination furnace.

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[*] Assistant Vice President, The W. S. Tyler Company.

[†] Consulting Engineer.

J. C. COSGROVE,* Johnstown, Pa.—I think we have talked too much about the economies or the economics of this matter of heat equipment. I believe the American people today are more interested in convenience, or in comfort, than they are in economies, and that there are a great many more available homeowners to whom to sell equipment than ordinarily we realize, because we look too much to the question of economies.

In almost every city in the eastern district gas burners and oil burners are being installed, and other apparatus of that kind, which cost a very much greater sum to operate than to burn coal, and it is being done entirely from a convenience point of view. I believe that is the basis on which most of these appliances are going to be sold. Certainly we want them to be as economical as possible, nevertheless they are going into homes no matter whether they are cheaper or more expensive to operate, if the convenience will justify; also, there must be taken into consideration the factor of time or work or attention that is required.

What we need in the solid fuel industry, in my judgment, is automatic heat for the homeowner. The anthracite people have gone a long way in that direction. The bituminous people have lagged behind; principally, in my judgment, because we have not had any centralized organization to follow it up to carry it out. I am not going to make any talk about the bituminous coal research, although I have been talking it for a long time.

There was some comment made on the question of power interruption and the drawback of power interruption to mechanical heating equipment. I think, with modern distribution of electrical power, that has practically ceased to be a factor. We have had experience with stokers and are now having some experience with automatic coal burners for which we have great hopes, and I cannot see that the ordinary interruptions of power will be any drawback to any of these methods.

The question of efficiency of heating equipment I have heard discussed many times, and it seems almost necessary to take some specified set of factors before starting such a discussion. I have heard the figures from the University of Illinois, wherein they take the efficiency as being the total amount of heat generated less what goes out the chimney at the top, on the theory that all heat made in the house is available heat for that house.

In Mr. Sherman's paper attention is called to that, but the statement is made that, of course, the heat ought to be heat that is able to be controlled or used where it is wanted. That I think is only partly true. Heat put into a house is advantageous, no matter where it is, except in so far as it may be excessive at some point. Heat that can be delivered at a certain point in the house is much more valuable than heat that has to find its way about, but the latter is not entirely lost because it may be generated in the basement and may go up through the floors and through the partitions, as the heat made in the basement gradually finds it way up through the house in any event. I have always found it advantageous to allow a certain amount of the heat to escape in the basement, in order to warm the floors of the first floor of the home.

There is a new movement under way in the automatic heating of homes with solid fuels; movement in the merchandising side to get the equipment to the householder for the minimum amount of initial expenditure. In other words, it is putting burning equipment on a lease basis. I understand that one of the large dealers in Cincinnati started a program of putting in stokers entirely on a lease basis, a combination of rental and an additional price for coal, so that the householder is going to pay in proportion to the amount of coal that he burns.

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* President, West Virginia Coal & Coke Co.

The company that I represent also has a plan of putting a burner for fuel in the home on a strictly lease basis.

I believe there is a lot to be done in the merchandising of equipment that has nothing to do with the economics of the equipment. In other words, getting back to my first statement that we are going to sell equipment because it is convenient, the householder is going to have it for convenience, not because it is going to save him money.

R. H. MORRIS,* Ansted, W. Va.—From a coal producer's standpoint, Table 3 is very interesting. In fact the paper throughout is interesting and so is the discussion, but being a coal producer, I am particularly interested in Table 3. In the last line it has hand-fired coal at a cost of $112 for the season and stoker-fired coal at a cost of $133 and fuel oil at $216 and gas at $207. That is a pretty interesting figure for one in the coal business, and one we like to see set up. The only thing is, we want to do a little more toward getting more coal used. I think the last gentleman who spoke hit the nail on the head when he said we should use more coal and more automatic heat.

Mr. Sherman has set up an item of ash removal. In producing coal we spend a lot of money to remove the ash before it goes into the furnace. Here we are spending money to remove the ash after it is burned. He also gives a figure of 10¢ a bushel, which on the coal here hand-fired and stoker-fired amounted to $3.45 and $5.10 for the season. That is quite an item.

The matter of coal research has been discussed a great deal. I wonder what we are doing on ash research. In other words, when you put in a stoker you get a cinder, and it seems to me something could be done with that ash perhaps to offset this expense here in ash removal.

Taking the hand-fired furnaces, I imagine in a good many cases the ash is filled up with old cans and some unburned coal and it would be rather difficult to do anything with it.

I happen to have a stoker heating my office and store, and this is the third year I have used it. The ash comes out in cinder form. It is an underfeed stoker. It seems to me something could be done with this ash.

The stoker no doubt has removed some of the disadvantages of soot. We have heard a great deal about smoke ordinances and so on. There is no question that the stoker has done a great deal to remove that trouble.

Now what can we do toward offsetting this expense of ash removal? Cannot we add ash research to our coal research?

R. PURDY,† Columbus, Ohio.—I think if you coal men would become evangelists and teach people how to use coal, you would find less coal in the alley and less coal on the neighbors' porches. I have a neighbor who thinks he has to have a clean grate all the time. The result is that he puts a lot of his coal on my front porch and a lot out in the alley.

As to the gas, I have a converter furnace. We operated that gas furnace two years at the same expense as we had previously operated our coal furnace, and I know how to fire a coal furnace without wasting coal out the chimney or in the ashes.

C. SCHOLZ.—I fear that one of the points not sufficiently stressed is the question of humidity. In my experience with hot-air heat, I found we were uncomfortable, my furniture cracked and every member in my household suffered from colds. I therefore installed a humidifier in my gas furnace, an ordinary V-shaped trough, but it did not evaporate enough water to raise the humidity sufficiently until I added a wick, about 30 in. long and 16 in. high, which has given very satisfactory results. My gas bills came down, my furniture stopped cracking and we are comfortable with the thermometer standing at 66°. Humidification is even more important in the home than it is in public buildings, because we spend more time in the home than elsewhere. The only way to convey this to the householder is through the salesmen of heating equipment and the coal salesmen.

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* General Manager, The Gauley Mountain Coal Co.

† General Secretary, American Ceramic Society.

J. Teegardin,* Columbus, Ohio (written discussion).—Mr. Sherman's paper has brought out in an excellent manner the important highlights of a domestic fuel comparison. The homeowner's problem is to weigh his heating cost against the convenience of operation.

One additional phase of the cost factor that might be mentioned under the heading of Fixed Charges is the cleaning of the equipment. This includes more than "Dirt"; it is a matter of obtaining efficient and satisfactory performance of the heating system. The amount of cleaning necessary with different coals varies widely. For instance, the Ohio No. 6 seam in the Hocking district produces a bituminous coal, which, compared to most other bituminous coals, has a very low hydrogen-oxygen ratio. The result is a fuel capable of burning with a very small soot formation. This condition, along with the flashy ignition provided by bituminous coals, makes Hocking a very desirable coal for domestic use.

A comparison of the cost of hand-firing Hocking egg coal, set up in the same manner as the other fuels given in Table 3, would be about as follows:

Thermal efficiency	50
Heating value dry, B.t.u	13,700
Heating value as received, B.t.u	12,900
Fuel used per year, tons	17.0
Cost of fuel per ton	$ 4.75
Cost of extra equipment	
Costs:	
Fuel	$80.90
Power	
Ash removal	5.95
Total fuel cost	$86.85
Fixed charges	
Total yearly cost	$86.85

(The delivered price of $4.75 per ton is based on prices of Sept. 1, 1933, in order that it may be comparable with the prices used by Mr. Sherman.)

The proper installation of a good thermostat or temperature regulator will often greatly increase the convenience of the heating operation. Where cost is important, the expense of heating the home used in this example with Hocking coal, showing an annual saving of $36 under the cost of the nearest competitive fuel given, is impressive.

It seems that a fertile field for the coal producers exists in urging the retail coal dealer to supply heat service to customers so that the domestic consumer may be freed of all worry and attention in connection with the heating of his home. Either a monthly service charge, or a fuel and service charge covering the coal delivery, firing, and ash removal would go a long way towards placing coal in a more favorable position with natural gas and oil.

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* President, New York Coal Co.

W. S. Blauvelt,* Troy, N. Y. (written discussion).—I am especially interested in Mr. Sherman's tabulation of cost factors and convenience factors governing the choice of fuel and equipment. As an engineer I agree entirely in this tabulation of factors, but two rather important factors have been left out:

1. The ignorance and gullibility of at least a considerable percentage of domestic fuel consumers;
2. The extraordinary plausibility and utter disregard for truth displayed by many vendors of various fuels and of a great variety of appliances.

Fortunately for those who try to solve the householder's problem of satisfactory heating effect at a minimum cost, "The majority of domestic users of coal are not likely prospects for automatic coal-burning equipment. They . . . cannot afford to buy equipment and must use the cheapest available fuels."

I have been greatly interested in the possibility of a low-cost method of processing coal. The most promising process in my opinion is the Wisner process. The process on which experimental work was done by the American Gas and Electric Co. at its Philo (Ohio) station a few years ago seemed to me particularly well adapted for treating the low-rank middle western coals that carry a high water content, and especially for those that have a low inherent ash, so that by commercially practical cleaning methods the ash in the coal as mined can be reduced to less than 6 per cent. Of course the commercial success of any such undertaking is dependent largely upon politico-economic factors, which may make a wasteful and socially undesirable practice profitable, while greatly handicapping any undertaking that conserves natural resources and could be used to satisfy human needs economically and efficiently.

Mechanical stokers for househeating appliances where the total annual consumption does not exceed 15 tons cannot be justified economically. Interest, depreciation charges, maintenance and power bill combined will in most cases greatly exceed the saving in the fuel bill resulting from the ability to burn a cheaper fuel. There is also, of course, somewhat greater liability to interruption of service with mechanical gadgets than there would be with a simpler appliance. An interruption of power service puts the mechanism out of business.

I have devoted a good deal of study to this question of an economical househeating appliance for use in the home where the total cost of heating is probably the determining factor as to both the appliance and the fuel. This class would I believe include about two-thirds of all the homes of the country. Satisfactory solution of the problem involves the following achievements:

1. The cost of the appliance installed should not greatly exceed the cost of standard appliances now in use.
2. Maintenance cost and freedom of interruption of operation for repairs should compare favorably with existing appliances.
3. Appliances should be self-contained; that is to say, not dependent upon uninterrupted power supply for successful operation.
4. There should be a wide range in the rate of burning fuel, that is pounds per hour, so that a fire could be kept ready to go with a total daily consumption of not over 10 lb. of fuel, while the same appliance could be operated successfully and efficiently when burning 10 lb. per hour.
5. The over-all efficiency (B.t.u. utilized divided by the B.t u. in the fuel fired) should be at least 75 per cent when the appliance is operated between 60 and 80 per cent of its rated capacity and should not be lower than 40 per cent when operated at 20 per cent of capacity.

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* Hudson Valley Fuel Corporation.

6. The rate of combustion should be controlled with a thermostat so that practically constant temperature is maintained at the place where the control is located.

7. Attendance should be required not oftener than twice in 24 hr. at the maximum rate of operation nor oftener than once in 24 hr. at not over 60 per cent of maximum rating.

8. The appliance should be so designed that it could readily be adapted to use either liquid or gaseous fuels with a high thermal efficiency and could quickly and at but trifling expense be altered to use solid fuel.

The combination of all the above specifications in a single appliance is likely to prove a little difficult. However, I feel that we are on our way toward the solution. The first experimental appliance would carry the fire all right when burning but 10 lb. of coke in 24 hr. and at that rate of combustion would supply 60 gal. of water, the temperature of which was raised 100° and which was drawn off at intervals about corresponding to the ordinary household requirements. At this rate, the over-all thermal efficiency would be about 40 per cent. When we ran capacity tests we had no trouble in burning 5.2 lb. of coke per hour with an over-all thermal efficiency of 78 per cent, even though there was 4 per cent carbon monoxide in the chimney gas.

When operated at a minimum rate with thermostat control this heater ran for a little over 74 hr., maintaining the temperature of the outlet water until the fire went out from lack of fuel, with no fuel added nor ashes removed after starting test.

The crude device we first experimented with gave such good results as to convince me that an entirely practical appliance could be designed that would approximately achieve all of the desired objectives. A second experimental heater was designed and is now being tested. We have operated it at rates varying from ½ lb. of coke to a little over 10 lb. of coke burned per hour. Maximum draft required for this maximum rate of combustion was 0.09 in. water column. The maximum operation temperature of chimney gas leaving the boiler was 288° F. Combustion gases still carry a little carbon monoxide, from 1 to 2 per cent, showing that secondary air ducts must be enlarged to give the desired results. The combustible in the ash refuse is very low; probably will not average over 3 per cent. It is safe to say that experiments to date have given some rather interesting results.

W. E. E. Koepler,* Bluefield, W. Va. (written discussion).—The element that is exerting the broadest influence on the domestic heating field is perhaps transportation and the attendant costs for the services rendered. In Columbus (Ohio) is one of the most complex freight-rate and transportation situations. This situation compares with that in St. Louis, where the retail coal dealers and much of the market for domestic coal has all but completely and permanently passed out. The whole blame for the loss of that market may not be traceable to transportation but much of it is and the same fate will happen to Columbus and other very attractive market cities, such as Cleveland, Toledo and all other Ohio cities, unless there is soon a complete overhauling of railroad transportation costs and services.

A second big factor in the loss of domestic heating business by the coal industry is the tendency of the distributors to meet competition on a price basis solely and utterly. Of course, the whole business structure has been absolutely panic stricken on a demoralized price structure, but this situation is notorious in coal. In Columbus, as a specific illustration: the retailers in coal are reported recently to have been discussing a small margin for cost and profit on all Ohio coals to meet the competition immediately threatening their existence on account of unbridled truck transportation. But they were seeking an agreement with the Federal Government which contemplated a much higher margin on Pocahontas and other high-grade coals. They were building up the business of the truckers in the main, unmindful that the inevitable result would be the extinction of the retail coal merchants because they were making the most attractive push in the very channel that the trucks could use to put them out of business. The retail coal trade can survive only by making quality products almost its entire stock and certainly by putting all of its sales and merchandising efforts behind the very best coals, accepting the cheap coal trade only by force of necessity.

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* Secretary, Pocahontas Operators Association.

These two conditions occur to me as presenting situations that make it impossible to pick out a given city at this time in which to conduct experiments with domestic fuels such as are discussed in the paper read by Mr. Sherman. In his paper, it seems to me, the price of Pocahontas lump coal is quoted higher than has been the prevailing level in the Columbus market in many years. Likewise there is a doubt that because of unsettled fuel conditions there might not have been used a higher, even if more theoretical, thermal efficiency figure than that assumed in the same paper. Efficiency is certainly something that will have to be looked forward to in the Columbus market as much as if not more than in other cities. Therefore it might be more in order to give them some efficiencies with coal to shoot at rather than to look down upon. The use of better coal and more modern hand-firing practice is much needed here.

We are not willing to accept the assumption of Mr. Sherman that the user of Pocahontas lump is the one who most likely will shift to automatic heating. We doubt this assumption because we naturally believe, and have had experience to support our faith, that there is but little more to be had in the way of convenience and comfort than can be had with a hand-fired, Pocahontas-fed, thermostatically controlled furnace. The Pocahontas Operators Association has an especially devised method of burning Pocahontas coal known as the "conical method" and except in severe weather twice a day is all the attention the furnace will need. Most automatic heating equipment will require that much attention and some should have a lot more attention, as accidents reported attest.

My own experience with an eleven-room house is that it requires only about 15 tons for an average season, while Mr. Sherman is allotting this tonnage for an eight or nine-room house in a climate not as severe as the mountain top at Bluefield, West Virginia.

Referring to the Cost Factors used in this paper, there might be some questions raised over the omission of the factor of labor but it need not give the advocate of coal concern if quality coals are used or modern combustion equipment employed. It has always seemed to me that the reformers of the heating systems have not gone back quite far enough in their efforts to improve conditions by giving the furnace itself and the transmission elements a lot of study and revision. Certainly, we are not making the most of the existing advantages of the constant heat supplied by coal as compared with the flashy performances of liquid fuels and gas with their intermittent heating and cooling of the home. This difference has a health factor that is important. It is alleged that substitute fuels leave cold floors in every house where they are employed. Coal with its steady performance permeates the whole house with healthy warmth.

There has occurred nothing to change our opinions in the Pocahontas group that one of the most needed things in the promotion of the coal trade either in competition with other fuels or for new uses is first and foremost more intelligent and more honest merchandising. We have done our part and will continue to do everything we can to protect our trade name so that consumers can get a coal suited to domestic heating by some method of identification which is practical; that is, the trade name route. If other coal producers and distributors would adopt policies along this line and stick to them there would be far fewer disgusted coal consumers turning to substitute fuels. We believe that these fundamentals of merchandising are the foundations of the N.R.A. program and we are encouraged to believe that we are given a new chance as well as a new deal in President Roosevelt's efforts to help the coal industry particularly.

J. HARRINGTON, Riverside, Ill. (written discussion).—I have always stressed the factors of convenience, cleanliness and automatic operation as dominating the domestic utility field. First cost is secondary in the minds of the great majority, although, obviously, this factor must not be lost sight of.

The first line of defense for the bituminous coal industry will be the domestic stoker that embodies all three of the foregoing qualities *plus* a low first cost. It can be done, and is going to be done shortly. And when this ideal combination appears on the horizon, it will be up to the coal man and his friends to rise up *en masse* and meet it halfway. Passive resistance may be all right in its place, but what the coal trade needs is a million 75's in the form of household stokers that will consume anything it has for sale. Then the householder can telephone his dealer for a load of coal, and the dealer can send over the kind he prefers to sell. You cannot get by on a national scale with a stoker that has to have its diet compounded by a combustion specialist, with terrific internal disorders the penalty for incorrect filling of the prescription.

Bituminous coal of any kind cannot only be handled dustlessly but can be burned smokelessly—and it can be burned *efficiently*. In 1907, I read a paper before the Western Society of Engineers, in which I laid down the principle that: "Combustion efficiency is independent of the *rate* of combustion." The day is not far distant when coal can and will be burned at rates from 10 to 20 lb. per hour with not less than 12 per cent of CO_2 and not over 10 per cent of combustible in the refuse. Then it will be up to some of the boilermakers to overhaul their antediluvian models so as to emit gases at temperatures of say 300° F. when operating at ratings demanded by practical operating conditions, instead of from 600° to 1000° common today.

And this, too, can and will be done. *Then* domestic heating with coal will take its rightful place in our scheme of things, and the producers, retailers, and consumers of a hundred or so million tons of the world's basic fuel will be happier, cleaner, and far more prosperous.

R. A. SHERMAN (written discussion).—Mr. Delamater has suggested that a comparison based on a furnace designed for gas would be fairer than on a conversion burner. This might well have been included, but it would have complicated the argument and it was desired to show the picture in the simplest manner with the fewest possible words. It is doubtful that the improved economy of a gas-designed furnace would offset the increase in fixed charges on a furnace with as low an annual load factor as a domestic furnace has.

Mr. Purdy has cited his own experience with the economy of gas. One wonders whether he had the same heating effect with gas as with coal, which is the only fair method of comparison and the one used in this paper. One frequently hears similar accounts but finds that the householder when starting to use gas, realizing that he was going to use a more expensive fuel, started to exercise economy by turning off heat in certain little used rooms or by maintaining a 70° rather than a 75° temperature. Often, too, weather-stripping or insulation has been done before using gas; these save gas, but similarly would have saved coal.

Mr. Scholz has made a helpful suggestion on methods to obtain higher humidities and the benefits to be derived. It is not generally recognized, however, that little economy in fuel results from humidification; the heat required to evaporate the water is approximately as much as the saving in heating the air to a lower temperature.

Mr. Blauvelt set up eight requirements for domestic heating units which appear sound, except that it would not seem necessary or even desirable to be able to burn liquid or gaseous fuels in the equipment designed for coal or coke. His progress with the coke-fired unit is promising and the final development will be awaited with interest.

Mr. Koepler believes that Pocahontas coal can be burned with as little attention as a stoker. It is believed that the twice-daily attention he cites is less than the average person will have to give to his fire and more than the average good stoker will require, at least with as good a coal as Pocahontas No. 3.

Mr. Harrington speaks of a stoker that will burn any coal. That is desirable, but to attain best efficiency with each coal the firing practice will undoubtedly have to be modified to suit the coal, as determined by experiment.

Mr. Cosgrove's comments on the excessive stress placed on economy in the sale of automatic firing equipment are in accord with the statements in the paper that most of those who are buying automatic stokers, oil and gas burners are purchasing cleanliness, convenience and comfort, and not fuel savers.

Anthracite as a Domestic Fuel

By Allen J. Johnson,[*] Primos, Pa.

(Columbus Meeting, October, 1933)

It is probable that the word "anthracite" was derived from the Greek preposition *an*, and the Greek word for *Thrace*, meaning, in combination, "toward or near Thrace." Aristotle, Theophrastus, Strabo and Pliny all mention Thrace, or Samothrace, in treatises upon stones and stone coals dating back to about 371 B.C.

In 1769 Obadiah Gore, a blacksmith of Wilkes-Barre, successfully burned anthracite in his forge, the first of millions of consumers of Pennsylvania anthracite.

The Anthracite Region

While anthracite is found in widely scattered locations throughout the world, 99 per cent of the total production of the United States originates within an area of 3300 square miles in eastern Pennsylvania, less than 500 square miles of which are underlain by workable coal beds.

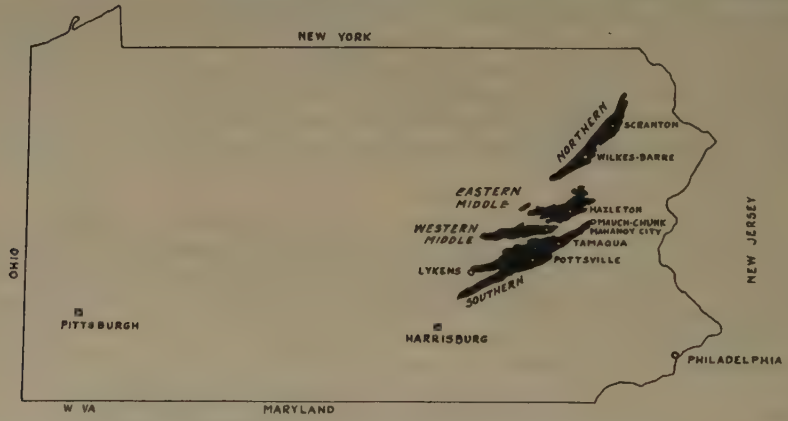

Fig. 1.—Anthracite coal fields of Pennsylvania.

Eminent authorities such as the United States Geological Survey agree that this Pennsylvania anthracite is the only true anthracite east of the Rocky Mountains. There is some graphitic anthracite in Rhode Island but it is not suitable for combustion; and coals from such states as Virginia and Arkansas, frequently sold as anthracite, are in reality semi-

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[*] Director, Anthracite Institute Laboratory.

anthracites. The latter differ from Pennsylvania anthracite in being considerably softer, with greater breakage, more free burning characteristics and higher volatile content, all of which make them more difficult to burn and to handle.

Pennsylvania anthracite occurs mainly in four irregular basins known as the Northern, the Eastern Middle, the Western Middle, and the Southern fields. These basins are the bottoms of downfolds or synclines in which the original coal beds were protected from the erosion of the glacial period that destroyed 94 per cent of the original deposits of the region.

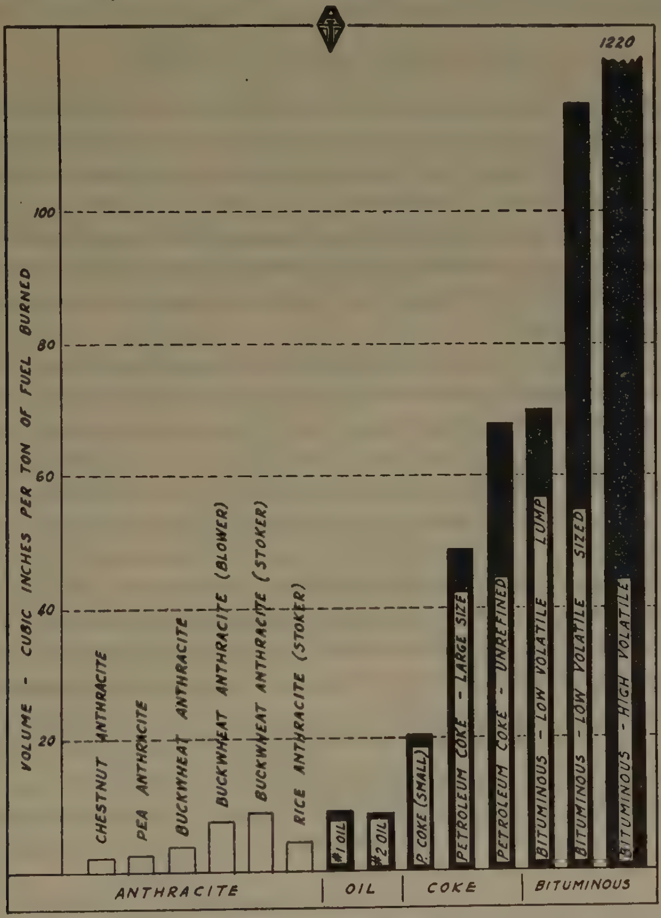


FIG. 2 —RELATIVE VOLUMES OF SOOT AND FLY-ASH PRODUCED BY VARIOUS FUELS.

Reserve Supplies

Despite the concentrated area of the Pennsylvania fields and the loss by erosion of much of the original supply, gigantic reserves of anthracite are still available. One reliable estimate places these reserves at ten billion short tons of recoverable coal. Estimates to determine the probable future life of this gigantic quantity of fuel are so vitally affected and extended by improved methods of production and improved and changed methods of utilization as to be highly problematical. It is certain, however, that ten billion tons of coal will be ample for generations to come, and that a fear of depleted supplies cannot properly be a valid reason for the use of substitute fuels.

Improved Preparation and Sizing

New and modern machinery provides a cleaner, better grade of fuel than ever before. A strong trade organization, The Anthracite Institute, representing over 95 per cent of all anthracite produced, is in operation to promote the interests of the industry as a whole. Through permanent divisions, the Institute maintains contacts with retail coal dealers, undertakes extensive research, and directs sales promotion, service and advertising. It has established and enforces standards of sizing and purity of preparation.

As a direct result of this determination on the part of the operators to supply fuel best suited to various industrial and domestic requirements, many commonly used tables of heat values and ash content have become obsolete during the past few years, and should be revised for modern engineering usage. This is particularly true of the smaller sizes, which originally were byproducts of the mines and have now become standard sizes.

The Anthracite Institute standards of sizing as shown in Table 1 are also being rigidly followed in order to insure a maximum of satisfaction. The following recommendations for the use of each of these sizes has been formulated by a committee of engineers representing the

Table 1.—*Standard Anthracite Specifications Approved and Adopted by the Anthracite Institute*

Round Mesh

	Test Mesh, In.		Oversize Maximum, Per Cent	Undersize, Per Cent	
	Through	Over		Maximum	Minimum
Broken	$4\frac{3}{8}$	$3\frac{1}{4}$	5	15	7.5
Egg	$3\frac{1}{4}$	$2\frac{7}{16}$	5	15	7.5
Stove	$2\frac{7}{16}$	$1\frac{5}{8}$	5	15	7.5
Nut	$1\frac{5}{8}$	$\frac{13}{16}$	5	15	7.5
Pea	$\frac{13}{16}$	$\frac{9}{16}$	10	15	7.5
Buckwheat	$\frac{9}{16}$	$\frac{5}{16}$	10	15	7.5
Rice	$\frac{5}{16}$	$\frac{3}{16}$	10	15	7.5
Barley	$\frac{3}{16}$	$\frac{3}{32}$	10	20	10

Effective April 1, 1931, revised March 18, 1931.

anthracite producing companies, in cooperation with the Anthracite Institute.

1. *Egg*, the largest size of domestic anthracite, should be used in fire pots having a diameter of not less than 24 in. and a depth of at least 16 in. The firing of this size in smaller furnaces, while justifiable in some instances, often results in an unnecessary ash-pit loss.

2. *Stove* coal is generally suitable for domestic heating plants where the fire pot is not less than 16 in. wide and 12 in. deep.

3. *Chestnut* coal is an ideal fuel for domestic heaters of any size where the draft is sufficient. A recent field survey conducted by the Anthracite Institute Laboratory disclosed a number of homes in which chestnut was being burned satisfactorily with an average draft of 0.065 in. of water. In several instances, drafts as low as 0.034 in. of water were employed without any heating difficulties. Chestnut coal is also a preferred fuel for kitchen ranges.

4. *Pea* coal can be used either in ranges or domestic heaters of any size where the draft is good. The field survey of the Laboratory showed that with an average draft of 0.07 in. of water, a mean over-all boiler and furnace efficiency of 56.7 per cent was being obtained with this fuel. This compares very favorably with other sizes of anthracite, and with other fuels.

5. *Buckwheat* is the smallest size that can be burned with natural draft. It is not recommended for use where the chimney is less than 50 ft. high or where the heating plant is naturally overloaded. Although it is possible to fire buckwheat upon large-mesh grates by carrying a layer of ashes below the active fuel bed, it is preferable to install fine-mesh grates for using this size. No. 1 buckwheat coal is frequently used in magazine-feed boilers, mechanical burners, and with forced-draft blowers.

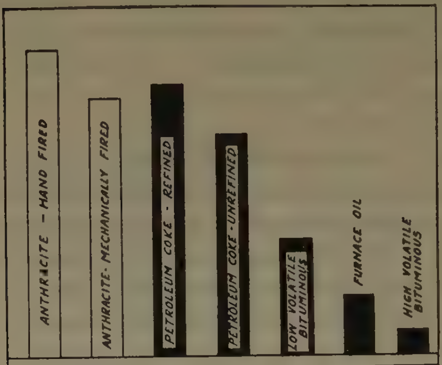
6. *No. 2 Buckwheat or Rice* anthracite in domestic heating is used only with mechanical stoking devices.

7. *No. 3 Buckwheat or Barley* has no application in domestic heating but is used extensively in manufacturing plants in connection with chain-grate stokers.

Cleanliness of Anthracite

The Encyclopedia Britannica states that "Some American cities enjoy a beautifully clear atmosphere owing to the smokelessness of the fuel." A complete absence of smoke and soot, owing to the extremely low volatile content of anthracite, has indeed become one of its best known characteristics. In order definitely to catalogue this advantage, tests were recently conducted in the Anthracite Institute Laboratory. During this investigation, the amount of soot, dirt and fly-ash passing up the chimney from 14 commonly used domestic fuels was determined by collection upon a 100-mesh screen and by subsequent actual measurement.

Each screen was allowed to remain in the stack for a period of one hour, during which time the boiler was operated in such a manner that the same amount of water was evaporated, irrespective of the fuel used. Every effort was made to test all fuels upon a basis of equal fairness. Fuels other than anthracite were fired in accordance with accepted best practice; and for oil a popular make of combination oil burner and boiler was purchased new and adjusted immediately before the tests by the manufacturer's representatives. The data secured, therefore, present a close approximation of the relative amount of solid material that might be expected to enter the stack with various types and kinds of fuel.



ANTHRACITE	DENSITY LB./CU.FT.
AVERAGE OF 3 SIZES HAND FIRED	14.4
AVERAGE OF 2 STOKER & 1 BLOWER FIRED SIZE	12.3
OIL	
AVERAGE OF 2 GRAVITIES	2.8
PETROLEUM COKE	
AVERAGE OF 2 SIZES REFINED	10.5
ONE SIZE UNREFINED	12 9
BITUMINOUS	
AVERAGE OF 2 SIZES LOW VOLATILE	5.5
ONE SIZE HIGH VOLATILE	11

FIG. 3.—RELATIVE DENSITIES OF FLY-ASH AND SOOT.

Materials of heavier density drop out of the atmosphere quickly and therefore are the least objectionable.

This investigation was analyzed from the following angles:[1]

1. The character of the material as related to its ability to permanently soil or injure fabrics.

2. The density of the foreign matter. (The area covered and air pollution is inversely related to the density of the material passing up the stack.) (See Fig. 3.)

3. The weight of the material emitted.

4. The volume of the material emitted. (See Fig. 2.)

Table 2 shows that fuels competitive to chestnut anthracite produced from 3 to 443 times as much solid matter by volume. The relative densities of stack solids from the various classes of fuels also shows that anthracite fly-ash is comparatively heavy and, therefore, that the small amount produced will have a tendency to fall back into the chimney, whereas the lighter materials from competitive fuels are more likely to be carried up the stack and deposited upon surrounding properties.

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[1] The following fuels were used in securing data upon relative cleanliness:

1. Anthracite of the hard white-ash type.
2. Anthracite of a medium burning type.
3. Anthracite, free burning.
4. Sinclair petroleum coke.
5. Broad Top bituminous, purchased as lump and broken to size especially for these tests.
6. Bituminous having a volatile of from 34 to 38 per cent.
7. No. 2 Solar oil of the Gulf Refining Company.
8. No. 1 Solar oil of the Gulf Refining Company.

TABLE 2.—*Relative Weight, Volume and Density of Solid Matter in Stack*
REFERRED TO CHESTNUT ANTHRACITE AS UNITY

Fuel	Nature of Solid Matter in Stack	Relative Volume[a]	Relative Quantity[b] (Weight)	Relative Inverse Density
Anthracite				
Chestnut	Hard granular	1	1	1
Pea	Hard granular	1.2	1.6	0.8
Buckwheat	Hard granular	1.4	2.3	0.6
Buckwheat (blower)	Fine granular	2.9	3.4	0.8
Buckwheat (stoker)	Fine gray powder	3.4	4.1	0.8
Rice (stoker)	Coarse gray granular	1.9	2.0	0.9
Oil No. 1				
Furnace	Flaked oily soot	3.4	0.6	7.6
Oil No. 2				
Furnace	Oily, sooty powder	3.3	1.3	2.5
Petroleum Coke				
Small	Sooty granular	7.3	10.0	0.7
Large	Sooty powder	17.5	11.3	1.6
Medium	Sooty granular	24.5	25.8	0.8
Bituminous				
Low-volatile lump	Sooty granular	25.2	14.1	1.8
Low-volatile sized	Sooty granular	42.0	20.2	2.1
High-volatile	Sooty, oily powder	443.0	45.7	9.7

[a] Relative volume based upon cubic inches per ton of fuel burned.
[b] Relative quantity based upon ounces per ton of fuel burned.

The material collected from the combustion of anthracite was, in all cases, a dry granular mixture of gray fly-ash and hard unburned coal. This ash did not contain any oily or sooty material and therefore would not, under any circumstances, stain or discolor even the chimney or roof of the home. The solid matter from other fuels was of a decided sooty, oily nature and was jet black, as contrasted with the gray-black to light gray shades of anthracite residue.

EVEN HEAT

The tendency of anthracite to produce an even quantity of heat over a long period of time is, undoubtedly, one of its greatest advantages as a heating fuel. Fig. 5 graphically illustrates this point, by showing the burning characteristics of three popular domestic fuels.

The essential parts of this test consisted of a commercial model of the Heatrola Space Heater having a fuel capacity of one cubic foot as measured on a plane level with the bottom of the firing door. The entire heater was placed upon Fairbanks platform scales.

The smoke pipes and all testing connections either were arranged in such a manner that they did not interfere with the periodical weighing of the heater and its contents, or were so connected that they could be disconnected for weighing.

By means of this apparatus connected to a standard 40-ft. brick and steel-lined stack, the combustion of one cubic foot of each of the various fuels was studied. The relative burning characteristics, the draft created as a natural function of the burning fuel, and the general combustion properties of the fuels in question were also determined.

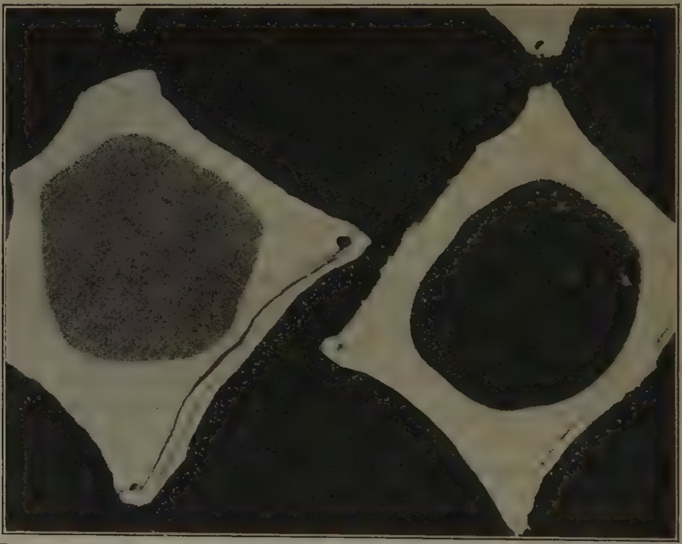


FIG. 4.—DENSITIES OF STACK MATERIALS FROM ANTHRACITE AND OIL.

All tests were both conducted and computed as new fire tests, in accordance with the corresponding code of the American Society of Heating and Ventilating Engineers. The following fuels were included in these tests:

1. Medium burning anthracite from the Western Middle field.
2. Koppers Philadelphia coke.
3. Broad Top bituminous coal.

These fuels were weighed periodically as they were burned in this heater. Comparative combustion results were obtained for complete firing cycles, and with the natural draft created by the fuel bed.

In no case did any fuel approach anthracite from a standpoint of the steady production of heat; whereas fuels that were either high in volatile matter, such as bituminous, or light in density, such as coke, both displayed a marked tendency to burn rapidly without the body or delayed action essential to a proper heating fuel.

Oil also was found to be inferior to anthracite from a standpoint of even heat, because of the fact that intermittent burners are used with a constant fluctuation in output.

SAFETY

Three factors are responsible for the fact that anthracite is one of the safest fuels used for heating purposes. Its nonexplosive properties, combined with a relatively high ignition temperature, preclude the possibility of accidental fires and explosions. The low percentage and nature of the volatile matter it contains eliminates the possibility of the production of soot, thus preventing chimney fires. The complete absence of spontaneous combustion in anthracite storage adds a final factor, thus making anthracite a truly safe fuel in all phases of its use.

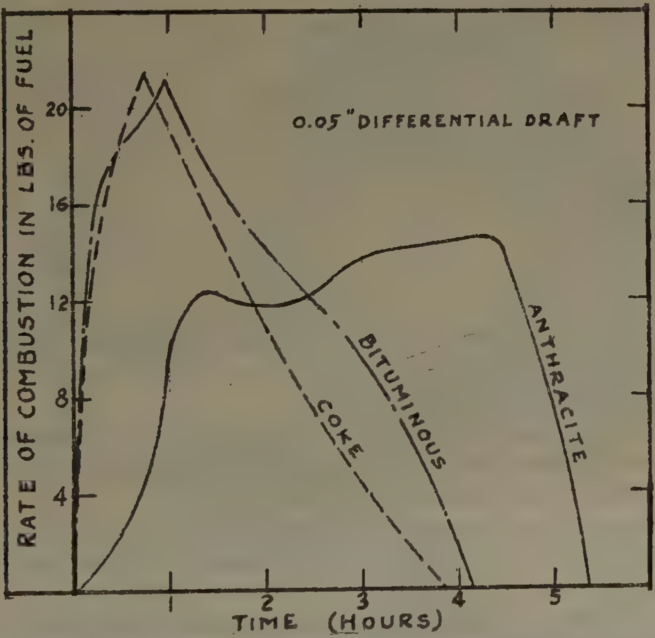


FIG. 5.—TYPICAL OPERATION CURVES, ANTHRACITE, BITUMINOUS AND COKE AS FIRED.

ANTHRACITE VS. COMPETITIVE FUELS

A series of investigations is now being conducted by the Anthracite Institute in order to determine the relative economic advantages of anthracite as compared with other competitive fuels. In the investigations that have been completed (byproduct coke, petroleum coke, furnace oil, and gas for service water heating) it has been conclusively shown that anthracite is a superior domestic fuel from several standpoints.

The heat value and analyses of the fuels used in these tests, as reported by Smith-Rudy Co., analytical chemists, are given in Table 3. All fuels were purchased in the open market, from retail fuel dealers selected at random, and were delivered from the regular stock of the retailer.

Anthracite vs. Coke

The analyses of byproduct coke were of particular interest in that an average of six samples showed anthracite to have a heat value of approximately 530 B.t.u. in excess of coke. It was also shown that the weight of anthracite averaged 55 lb. per cubic foot, whereas coke weighed only 28¼ lb. for an equal volume. The fusion point of anthracite ash was considerably above that of coke, whereas the ash contents of the two fuels were practically the same.

TABLE 3.—*Heat Values and Analyses of Fuels Tested*

	Pennsylvania Anthracite (Average of 3 Samples)	Byproduct Coke (Average of 3 Samples)
Heat value, B.t.u	13,708	13,168
Ash content, per cent	8.70	8.58
Fusion point, deg. F	2,770	2,516
Pounds per cu. ft	55	28¼

One of the principal points of superiority for anthracite was the production of even heat over a long period of time, and the materially reduced firing attention which it necessitated.

As shown in Fig. 6, hard coal required only one-half to one-quarter the attention of coke, according to the conditions under which the fuel was burned.

The "New-Fire" method of testing sponsored by the American Society of Mechanical Engineers was used throughout, with fuel-bed conditions as indicated in Table 4. The apparatus used was identical with that described on page 365.

Unusual conditions of firing, such as heaping the fire pot, were included in this comparison. It is of interest to note, however, that in such cases the normal output of the boiler was proportionately reduced, irrespective of the fuel.

The fact that anthracite can be banked twice as long as coke proved to be another major advantage disclosed by these tests.

Petroleum Coke

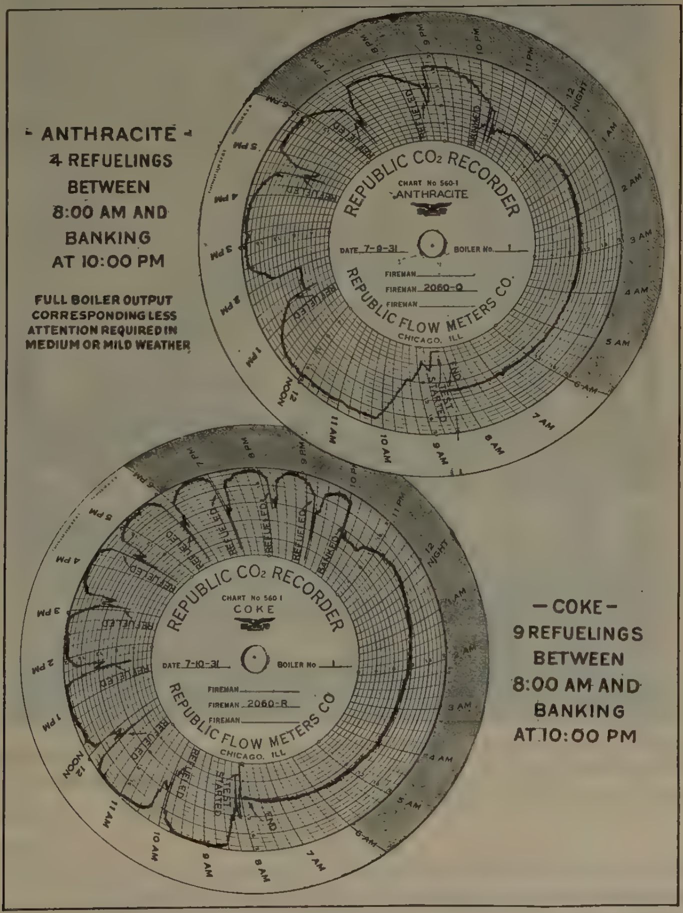
As compared with petroleum coke, and other ashless solid fuels, an additional advantage for anthracite was found to be that it was not

TABLE 4.—*Number of Firing Periods per Day*[a]
Exclusive of Starting Fire in Morning and Banking at Night

Operating Condition	Anthracite	Coke
Fuel level with bottom of door, dampers ¾ open	1	4
Fuel heaped to top of door, dampers ¾ open	1	2[a]
Fuel level with bottom of door, dampers open[b]	4	9

[a] While the practice of heaping coke in the furnace is shown to decrease the attention it should also be noted that the output of boiler was reduced to 75 per cent of its normal value.

[b] Illustrated in Fig. 6.



- ANTHRACITE -
4 REFUELINGS
BETWEEN
8:00 AM AND
BANKING
AT 10:00 PM

FULL BOILER OUTPUT
CORRESPONDING LESS
ATTENTION REQUIRED IN
MEDIUM OR MILD WEATHER

- COKE -
9 REFUELINGS
BETWEEN
8:00 AM AND
BANKING
AT 10:00 PM

FIG. 6.—RECORD OF ATTENTION REQUIRED BY ANTHRACITE AND COKE (FULL STACK DRAFT).

nearly so destructive to the grates of the furnace. As an example of the destructive action of petroleum coke, a Pennsylvania realtor has included the following clause in all of the leases of his company: "It is agreed that the lessor is not responsible for grates in the furnace if the lessee burns petroleum coke therein." Fig. 7 illustrates this condition by showing the lack of protective ash on the grate bars.[2]

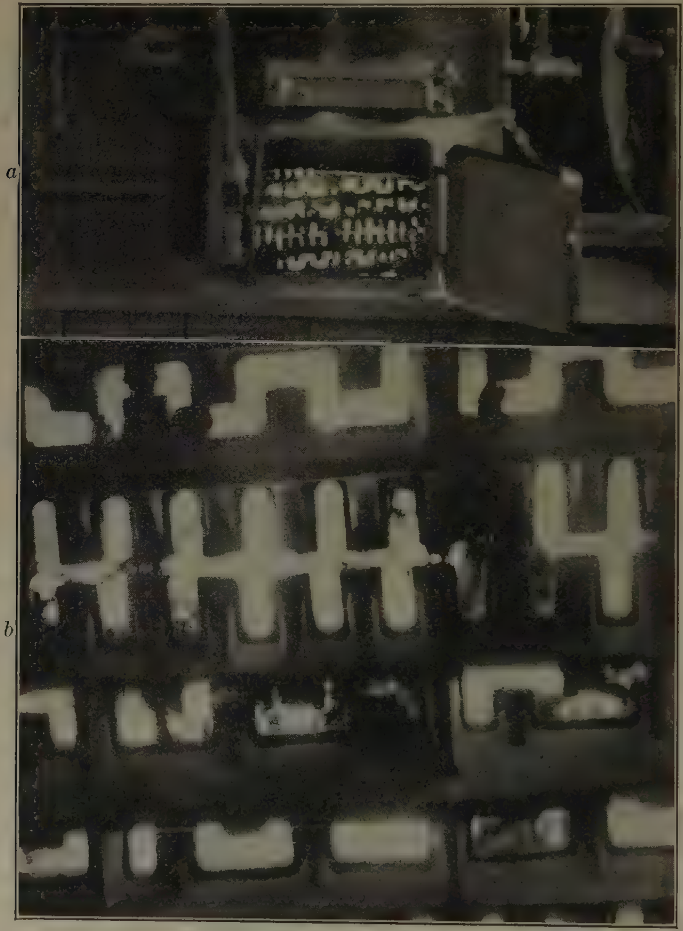


FIG. 7.—(*a*) VIEW WITH MIRROR IN ASHPIT WITH BURNING PETROLEUM COKE, SHOWING LACK OF PROTECTIVE ASH AND INTENSE HEAT ON GRATE BARS. (*b*) VIEW OF GRATE BARS FROM BENEATH ASHPIT.

Furnace Oil

An investigation of anthracite and oil was conducted upon one of the nationally advertised and best selling oil-burning boilers, installed and

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[2] The fuel used for securing these pictures was sold in Philadelphia as C99. It is reputed to be a high grade of petroleum coke and is being especially recommended for domestic heating.

TABLE 5.—*Gallons of Oil Equivalent to Net Ton of Anthracite*
With Various Types of Equipment and Operation

	Oil Conversion Unit	Oil Boiler Burner Comb.
At average operating rating[a]		
Chestnut hand fired	185	185
Buckwheat stoker fired	169	169
Buckwheat magazine-feed boiler	173	173

[a] This rating should be used for all seasonal comparisons.

TABLE 6.—*Comparison of Operating Costs*[a]

	Oil Conversion	Anthracite	
		Hand Fired	Magazine Feed
Fuel costs			
Oil @ 6 ¢ per gallon	$111.00		
Chestnut @ $13 per ton		$130.00	
Buckwheat @ $8.50 per ton			$ 89.00
Gas @ $1.00 per M	3.50	00	0
Electricity @ 6 ¢ per kw-hr	10.00	00	0
Fixed charges			
Depreciation 10 per cent	50.00	[b]	20.00
Interest and repairs 6 per cent	30.00		12.00
Total	$204.50	$130.00	$121.00
Annual saving in favor of anthracite		$ 74.50[c]	$ 83.50

[a] Based on residence consuming 10 tons of chestnut anthracite per year.

[b] It is assumed that this is a conversion job and that the owner has a boiler. Even if this is not so, fixed charges would be the same against both oil and anthracite.

[c] Savings will be increased in direct proportion as fuel used exceeds the equivalent of 10 tons.

adjusted to a point of maximum efficiency by service representatives of the manufacturer. While every effort was made to secure fair and representative tests, it should be realized that there is a wide variation in the performance of various oil burners, and that the results described herein might thus be changed with a different burner or under different test conditions.

It was found in these tests that at the point of average demand upon the heater, namely, 32 per cent of its maximum output, 185 gal. of oil would be required to replace a net ton of hand-fired chestnut anthracite (Table 5). It was further shown that a home consuming 10 tons of chestnut anthracite could normally be heated for $130; whereas, with a conversion oil burner, $205.50 would be needed to cover the cost of the fuel, gas, electricity, depreciation at 10 per cent and interest and repairs (Table 6).

A further limitation to the use of oil for domestic heating is the inability of designers to produce an oil burner that can be modulated; that is, one in which the fuel feed and heat production can be adjusted to suit the existing weather conditions. Because of this limitation, it is necessary to operate an oil burner at a fixed rate, depending upon its shutting off entirely if producing any less heat than its maximum capacity. Since the maximum is demanded during less than one-third of the time, it is apparent that for two-thirds of the heating season the burner is supplying heat intermittently with corresponding fluctuations in the room temperature. As no thermostat can be designed to respond instantaneously, this sometimes results in discomfort and is always inferior to the steady, even heat of a properly controlled anthracite fire.

Table 7.—*Comparative Costs*

ANNUAL COST OF HEATING WITH ANTHRACITE AND OIL

Oil		Anthracite Annual Cost		
Price per Gallon, Cents	Annual Cost	Price per Ton	Hand Fired	Magazine Feed[a]
4½	$176.75	$ 7.00	$ 70.00	$114.75
4¾	181.50	7.50	75.00	112.50
5	186.00	8.00	80.00	116.75
5¼	190.50	8.50	85.00	121.00
5½	195.25	9.00	90.00	126.25
5¾	200.00	9.50	95.00	131.50
6	204.50	10.00	100.00	136.75
6¼	209.00	10.50	105.00	142.00
6½	214.00	11.00	110.00	147.25
6¾	218.50	11.50	115.00	152.50
7	223.00	12.00	120.00	157.75
7¼	227.75	12.50	125.00	163.00
7½	232.50	13.00	130.00	
7¾	237.00	13.50	135.00	
8	241.75	14.00	140.00	
8¼	246.25	14.50	145.00	
8½	251.00	15.00	150.00	
8¾	255.75	15.50	155.00	

[a] Buckwheat or pea coal.

PRICE FOR WHICH OIL MUST BE PURCHASED TO EQUAL COST OF HEATING WITH ANTHRACITE (HAND FIRED)

Chestnut Anthracite, per Ton	Oil per Gallon, Cents
$ 9.00	[a]
9.50	⅛
10.00	⅜
10.50	⅝
11.00	⅞
11.50	1⅛
12.00	1½
12.50	1¾
13.00	2
13.50	2¼
14.00	2½
14.50	2¾
15.00	3

[a] Fixed charges alone greater than entire cost of heating with anthracite.

The president of a leading oil-burner company recently estimated that this ignition cycle may be repeated as often as 12 to 24 times per day, or 2500 to 5000 times during the heating season. Each of these cycles implies a variation in room temperature in addition to placing considerable strain upon ignition equipment.

City Gas

Extensive campaigns are being waged in a number of eastern states to induce homeowners to heat their buildings with gas. The anthracite salesman can concede that gas is convenient and even that its furnace efficiency is higher, but the cost is usually so excessive that its use is almost prohibitive. Based upon a 15 per cent increased efficiency, 540 B.t.u. of gas at a retail price of $1.00 would cost as much with all allowances for the increase in efficiency as would anthracite coal at $36.75 per net ton. Table 8 shows other equivalent costs based upon the same efficiency but with various initial gas rates.

TABLE 8.—*Relative Cost of Anthracite and Gas*

Cost of Gas per 1000 Ft.	Equivalent Cost of Anthracite per Net Ton
$0.60	$22.00
0.70	25.75
0.80	29.50
0.90	33.00
1.00	36.75
1.10	40.50
1.20	44.00
1.30	47.50

SERVICE WATER HEATING

Anthracite is also extremely economical for heating service water, as compared with city gas. An investigation by the Laboratory was conducted as based upon two sources of information, namely:

1. Actual field tests in which the exact data in connection with the consumption of both gas and anthracite were accurately measured by Laboratory engineers.
2. Laboratory tests in which the fuel consumption of bucket-a-day stoves was determined during both running and banking periods.

The results of these two radically different sources of information are in remarkable accord in showing:

1. That an anthracite "bucket-a-day" stove can be operated at the remarkably economical cost of 10 to 12¢ per day for any water consumption up to 127 gal. (100° rise water) of hot water per day.
2. That an adequate supply of hot water is available with anthracite without additional cost or increased manual attention.

3. That a 16-hr. banking period with anthracite consumes only 5¢ worth of coal ($12 per ton).

4. That the cost of heating water with anthracite is only ⅓ to ⅕ (average ¼) the cost of heating the same amount of water with gas.

Convenience

We who are actively engaged in research in connection with the utilization of anthracite are thoroughly convinced of its inherent superiority as a domestic fuel. As its safety, cleanliness, even heat and economic advantages have been outlined in detail, the one remaining factor of importance is that of convenience.

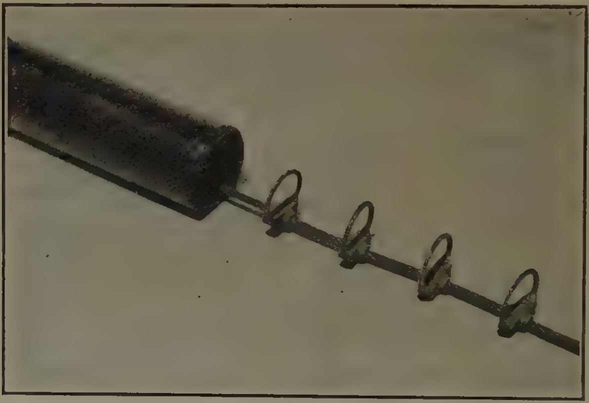

Fig. 8.—Detail of anthracite institute ash conveyor.

As a hand-fired fuel anthracite has been shown to require less attention than other solid fuels, whereas in competition with liquid and gaseous fuels the availability of automatic and semi-automatic equipment is making automatic heating with anthracite an economical reality.

Dependable stokers and magazine boilers are available and may be depended upon for supplying heat without manual attention for periods of from 24 hr. to several days (depending upon the rate at which the boiler is operated). During one series of tests in the Laboratory a stoker was set at a rate that approximated the heat demand of the building and was left unattended over a holiday, without thermostatic control. Although the outside temperature dropped 23° during this period, the stoker maintained the building temperature at 70° F. during the entire period.

In several instances, coal bins have been designed to feed directly to the stoker worm, thus providing complete automatic heat in a non-mechanical manner.

Conveying equipment to fill stoker hoppers from the coal bin, and special ash nozzles to fill seven large cans in turn, are available as standard accessories. When these are used in conjunction with thermostatic control, the continuous occupancy of the house bears but little relation to its temperature. With anthracite, the relegation of such duties to robots does not in any way increase the fire hazard.

Furthermore, the Anthracite Institute Laboratory, spurred on by absolute necessity, is actively engaged in providing weapons for the battle of fuels which is already being waged, and which we must enter with adequate equipment.

The century-old problem of ash removal has been solved by the Laboratory through the development of a mechanism of which the major part is a slowly moving chain by which the ash may be removed to any point and stored in any desired quantity. The installation of this conveyor is not as complicated as an outside oil tank, and its initial cost is less.

With such a definite solution to what is perhaps one of our most perplexing problems, the objections of the architect, builder and housewife to the use of hand-fired anthracite can be overcome to a point where the position of the industry will be greatly strengthened.

The second problem from a standpoint of popular appeal is the development of a compact unit that can be offered as an answer to the consumers' demand for automatic heat.

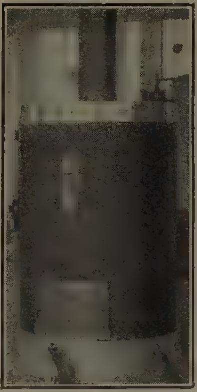
FIG. 9.—BOILER UNIT.

This desire for a modern heat machine for all up-to-date homes, in competition with compact oil-burning and gas-burning units, led to the first completely automatic domestic solid-fuel furnace ever perfected. This machine, which was developed by the Anthracite Institute Laboratory, combines effortless firing, temperature control, outside ash disposal, and hot-water supply in one single-motored unit, and is designed for convenient connection with air-conditioning or humidifying equipment.

The boiler developed for this purpose is of such an efficient design that ratings have been consistently secured 33 per cent greater than those of the latest oil-burning boilers, even though the unit is considerably more compact and is devoid of the complicated mechanism and controls of an oil-burning boiler of the same type.

When placed upon a production basis, the appearance of the unit as finished in chromium and black enamel, with modernistic lines, should certainly give anthracite the reputation of having the most modern of heating equipment.

The inherent advantages of anthracite, combined with these and similar types of equipment, are certain to increase the popularity of anthracite. Its continuance as the standard of domestic fuel for the comfort, safety and convenience of coming generations is thus assured.

DISCUSSION

(Charles E. Lawall presiding)

E. T. Conner,* Scranton, Pa.—Mr. Johnson has done in his experimental work what I have long hoped would some day be accomplished. He has so fully covered the subject that there is very little that I can say. He omitted one item, however; the matter of cleanliness in large cities. Most cities have smoke ordinances, which are more conspicuous by not being enforced than by observance. It has been computed that New York City, where much bituminous coal and oil are burned, loses $96,000,000 annually as a result of the damage done by soot particles, and that certain monoxide sulfuric acid-arsenic compounds and tar products are continually being emitted from the boiler plants where these fuels are burned, causing injury to health as well as to property.

N. G. Alford,† Pittsburgh, Pa.—In regard to Mr. Johnson's chart showing rates of burning for coke, bituminous and anthracite, it seems that the more rapid acceleration of both bituminous coal and coke makes them more responsive to delicate thermostatic control, without the use of auxiliary drafting, which I suppose would be necessary in the case of anthracite to get the same response. I would like to hear Mr. Johnson's comment on that point.

J. R. Campbell,‡ Pittsburgh, Pa.—The author has stated, on page 368, that the average of six samples was 530 B.t.u. higher than coke. There may have been different kinds of coke. I think it would be very fair to state the proximate analysis of the anthracite and coke in comparison.

It is almost unbelievable that a good quality of coke would be lower in B.t.u. than the same quality of anthracite. Will the author justify his figure on the comparative ash and clarify that point?

A. J. Johnson.—I realized that in presenting this paper I should have to go light on bituminous or not mention it at all. I tried to go very light.

As a matter of fact, I lived for three years in Indianapolis so that I should know that high-volatile bituminous is used in homes. However, we in the anthracite region have not tried to recommend anthracite for the strictly bituminous territory. We do not say that anthracite should be used tomorrow in every home in Columbus or Indianapolis. When we speak of anthracite and bituminous, we are speaking of regions in which anthracite is commonly used; in other words, what we commonly refer to as the anthracite-burning territory. In that territory, I think you will agree with me that the high-volatile bituminous has a very small domestic foothold.

As far as the analysis of the fuels is concerned, the anthracites that we analyzed average well over 13,000 B.t.u.

J. R. Campbell.—What is the percentage of ash?

A. J. Johnson.—The anthracite averaged—I am taking a chance in giving the exact figures—13,400 B.t.u. When I mention a figure of that kind, as I said in the paper, there will probably be hands raised in horror. But we are getting it. It is being produced every day. Such anthracite comes out of the anthracite region, and it is nothing like the figure used 10 years ago in so far as ash content and B.t.u. are concerned. It is an entirely different fuel. It is better prepared; it is prepared with modern machinery, and is a real fuel. Today 13,400 B.t.u. is by no means an impossible specification for any operator to meet.

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* Mining Engineer.

† Mining Engineer, Eavenson, Alford & Hicks.

‡ Bituminous Representative, Koppers Rheolaveur Co.

The cokes we bought were purchased in the open market in the vicinity of Philadelphia. One of them was the Philadelphia Koppers coke. One was a Rainey-Wood coke. One was a Philadelphia Electric Chester coke. There were three other cokes shipped in from surrounding territories. The average analysis of those six samples of coke was about 12,800 B.t.u. We have nothing further upon which to base our statements.

Similarly on our high-volatile bituminous, I will grant that there are all kinds of bituminous in the world. I think probably that is one of the major problems of the bituminous operator. We went to the coal yards and asked the dealer for a certain amount of anthracite and a certain amount of bituminous. He sold us both. We asked for high-volatile bituminous, and he sold us that. We did not make any effort to secure any special kind.

Frankly, I think the bituminous people would do well to place a lower volatile bituminous in this territory if that which is being sold is too high. Probably that is where the fault lies.

In so far as rapid acceleration is concerned—I think that was the point that Mr. Alford raised—it is true that the more rapid the acceleration, the more rapidly the thermostat will respond, but it is also true that the thermostat would have to respond a certain additional number of times because of the fact that the acceleration equals the deceleration. In other words, while the thermostat comes on more quickly it also goes off more quickly with a fuel having increased rapidity of response.

E. L. **Buller**,* Scranton, Pa.—I would like to ask Mr. Campbell a question. I have here the analysis of anthracite that we tested in our chemical laboratory, and which was also put under competitive fuel tests in our equipment laboratory, located on our property. On the analysis of anthracite you brought out the point that you did not see how two fuels of equivalent ash, comparing coke and anthracite, could have a difference in B.t.u. value.

J. R. **Campbell**.—A difference of 500 B.t.u.

E. L. **Buller**.—Here is one with a difference of from 245 to 250, or in that neighborhood. The volatile is 7.3. The carbon is 82.0. The ash is 10. The sulfur is 0.75. The B.t.u. value is 13,560. That is on a dry basis.

The volatile in the coke is 1.2, made on exactly the same basis, the dry basis. The comparison is fair. The carbon is 90.8. The ash is 8 per cent. The sulfur is 0.70. The B.t.u. value is 13,310. That was a New England coke, made, I believe, at the Koppers coke plant, in the vicinity of Boston.

Member.—Was the coal burned on the dry basis?

E. L. **Buller**.—Yes.

Member.—Was the inherent moisture taken out?

E. L. **Buller**.—That is a question I cannot answer, but the inherent moistures I have run have indicated that they are higher on coke than on coal.

J. R. **Campbell**.—Will you please give us the composition of the coal?

E. L. **Buller**.—I have compositions of a large number of veins in the anthracite field. They range from a minimum of 0.38 to a maximum of 1.25.

In connection with these coke tests, Mr. Campbell, going into the fuel test side of it, the boiler was operated to deliver, as nearly as it was possible to determine beforehand, an equal load, when both the coke tests and the anthracite tests were conducted. The operation was set up so that we could generate about 500 sq. ft. of steam radiation, which was approximately 72 per cent of the boiler's maximum rating. We ran a number of tests, first at different rating points to get the most efficient operating point of the boiler; i.e., the most efficient load point. The coke tests were run for a series of about 270 days, approximating, as nearly as possible, the conditions met during a complete heating season.

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* Combustion Engineer, Hudson Coal Co.

Similar tests were run on an anthracite boiler of exactly the same design, the same square feet of grate surface, and so on. The anthracite boiler was operated to generate approximately the same number of square feet of steam radiation.

The coke boiler started out at an efficiency about 2 per cent higher than that of the anthracite boiler, but over a period of a heating season the efficiency dropped down to between 15 and 24 per cent below the anthracite efficiency, owing to flue-dust accumulation in the heating surfaces of the boiler. We also found that, owing to the lower density of the ash made from the coke, the natural draft carried considerable amounts of it up into the heating surfaces, and the coating of soot on these heating surfaces naturally decreased the efficiency of the boiler. That is as fair a way as I know of comparing the two fuels.

We burned bituminous coal in a test boiler exactly like that used in the anthracite and coke tests and found that over a heating season the efficiency dropped off materially, because the soot and the unburned carbon particles covered the heating surfaces and caused a decrease in efficiency. Probably we can be accused, as anthracite engineers, of not knowing how to burn bituminous coal efficiently. At any rate, we gave it the best possible break. We fired it as nearly as possible to conform with the best combustion practices as to excess air, etc. The efficiency died down toward the end of the heating season, which unfortunately does not give the householder the full benefit of the fuel that he has purchased at a time when the house is hardest to heat: that is, in the spring of the year, when we may have alternately warm and cold days.

The company I represent has done considerable work with automatic equipment, through the New England states in particular and in the vicinity of Philadelphia and New York. We have made quite an extensive survey of the consumer market for automatic equipment. We have found, almost without exception, that the consumer is interested in the convenience he can obtain from automatic anthracite-burning equipment, while the cost is secondary, provided it is not out of all human reason. In other words, the consumer who is equipment-minded, or the consumer looking for equipment, is going to have something automatic, providing he does not have to pay an exorbitant price for it. The convenience that goes with that automatic equipment is relative in some cases. That is, in some cases the consumer may want a thermostat and an ash remover; apparently he does not object to shoveling on coal. Usually he is a consumer of 10 tons or less. The larger customer (10 tons or more) wants heat that is absolutely automatic and will pay a fairly good price for this equipment.

P. L. MIKESKA,* Schenectady, N. Y.—I have been connected more or less with apparatus for burning all kinds of fuels; oil, gas and coal, both hard and soft. Mr. Johnson's paper mentions soot and dirt on the screen, which I imagine was with the conversion oil burner, was it not?

A. J. JOHNSON.—Yes.

P. L. MIKESKA.—The more modern oil burners are combination burner-boiler units. It would be interesting to have such a test as that made and added to this report. Also, there are several makes, and to get a good picture several makes should be checked.

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* General Electric Co.

The modern oil burner-boiler units supply domestic hot water the year round, therefore the boiler water is at all times at a temperature higher than, say, 170° F. So if the thermostat calls for heat, there is almost immediate response, and if the burner-boiler unit is properly adjusted to the load in the home, it can be operated at a very high efficiency without the overshoot.

The thing I am trying to indicate is that, even with oil, you might get the Appalachian type of curve of which you spoke.

Uses of Coal in the Ceramic Industry

By H. E. Nold,* Columbus, Ohio

(Columbus Meeting, October, 1933)

The raw materials of the ceramic industry are mostly clays. This raw material is ground, water is added and the mixture pugged into a moist, plastic, rather stiff mass. From this mass the desired units of manufacture are made (bricks, hollow building tile, sewer pipe, etc.). These units are dried; then placed in kilns and heated until the desired degree of vitrification is obtained.[1] For firing ceramic ware, two distinct types of kilns are used; namely, periodic and continuous tunnel.

Periodic Kilns

Periodic kilns are those ordinarily seen in a brick plant. They are either round or rectangular. The raw, dried, formed ware is piled or set in the kilns to depths frequently exceeding 10 ft. The kiln is heated until the ware is matured, then gradually cooled sufficiently to allow men to remove the ware; then the cycle is repeated. The time duration of the cycle may be more than two weeks.

In order that heat can be distributed uniformly throughout the pile or setting, the units are placed so as to leave open spaces or flues throughout the pile for the passage of the hot gases. Each kiln is provided with a number of fireboxes (usually 10 or 12) spaced equally in the outside wall. The fireboxes may be equipped with flat or inclined grate bars or they may have no grates at all (dead bottom). Down-draft kilns are most used. In these the hot gaseous products of combustion from the fireboxes are deflected upward by bag walls just inside the fire zone. From the top of the kiln the gases are drawn down through the setting, off through holes in the kiln bottom, and conducted to a stack or stacks. In up-draft kilns the fireboxes are usually set somewhat lower. The hot gases are drawn inward and upward through the ware and pass off from the top. Up-draft kilns find their greatest use in the firing of common brick. In either case the problem is to maintain, as nearly as possible, a uniform temperature throughout the ware being fired, and to gradually raise the temperature of the entire mass to the point necessary to mature the ware.

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* Professor of Mine Engineering, The Ohio State University.

[1] This forming procedure is not followed exactly for all products. Some products are pressed from the ground clay without pugging; others, such as cups, pitchers and some sanitary ware, are cast from a clay slip. In all cases, the dry formed product is fired in kilns one or more times.

Heat is carried to the ware by convection (moving hot gases) and by radiation from the incandescent particles of carbon (soot) and fly-ash carried by the rapidly moving gases. These incandescent particles appear as a long flame extending throughout the ware and at times showing out at the top of the stack. On account of this luminous flame, it has been rather generally supposed that the complete burning of the gaseous products of combustion is delayed until the gases are in the midst of the pile of ware. The superiority of high-volatile coal for kiln firing was attributed to this supposedly delayed combustion. Work done by the United States Bureau of Mines[2] shows that the combustion of the gases from the fuel bed is very rapid and, under normal conditions, is completed within a few feet of the fuel bed. The authors state that the incandescent particles perform an important function in the distribution of heat within the kiln: "They are heated to incandescence by the heat of combustion at the fuel bed and carry this heat within the setting where they deliver it to the ware by radiation."

HIGH-VOLATILE COALS IN KILNS

The desirability of high-volatile or long-flame coal is well recognized by the industry and is further emphasized by W. E. Rice, who lists the characteristics of ideal coal for the ceramic industry as shown in Table 1.[3] Noncoking coal is desirable.

TABLE 1.—*Characteristics of Ideal Coal for Ceramic Industry*

	PER CENT		
Moisture	2	Sulfur	not more than 1 per cent
Volatile matter	36	Calorific value	14,000 B.t.u. per lb.
Fixed carbon	56	Softening temperature of ash	2600° F.
Ash	6	Color of ash	white
	——	Size	2 × 4 in.
	100		

Mr. Rice then lists the properties of six bituminous coals found satisfactory for kiln firing as shown in Table 2.[4]

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[2] G. A. Bole and others: Problems in the Firing of Refractories. U. S. Bur. Mines *Bull.* 271 (1927) 150 to 156, 159, 160.

[3] W. E. Rice: Properties of Coal Which Affect Its Use in the Ceramic Industry. *Trans. A. I. M. E.* (1932) **101**, 247.

[4] Reference of footnote 3.

Table 2.—*Properties of Coals for Kilns*

	Coal Number[a]					
	1	2	3	4	5	6
Proximate analysis:						
Moisture, per cent	2	3.6	7.5	4.4	3.3	7.2
Volatile matter	36	35.6	32.5	40.9	38.7	31.6
Fixed carbon, per cent	56	54.6	57.0	45.4	50.9	50.2
Ash, per cent	6	6.3	3.0	9.3	7.1	11.0
Sulfur, per cent	1	1.0	0.8	3.1	2.5	1.0
Fuel ratio, fixed carbon/vol. matter	1.56	1.53	1.75	1.11	1.32	1.59
Calorific value, B.t.u. per lb	14,000	13,540	12,500	12,510	13,430	11,790
Softening temperature of ash, deg. F	2,600	2,460		2,730	2,340	2,190

[a] No. 1 is the ideal coal that has been given in this paper.

No. 2 is the average of 80 coals of the United States that are considered good coals for kilns.

No. 3 is representative of a coal of the Appalachian Region of the eastern province that is said to be one of the best in the United States for kilns. This coal has not the hardness demanded of the ideal coal, but it does not coke in the furnace, and the softening temperature of the ash is high.

Nos. 4 and 5 are coals that are used to fire bricks made of clay that is not damaged by sooting, reducing atmosphere or sulfur. The fuel ratios of both are low and they produce dense smoke in the kiln.

No. 5 cokes quickly in the furnace, hence is more satisfactory in wide than in narrow furnaces.

No. 6 is a coal from the eastern region of the interior province. This gives satisfactory results in kilns of ware that is not affected by the ash. It does not coke in the furnace. Because it forms clinkers easily, it can be burned in wide dead-bottom furnaces better than in narrow furnaces or on grates.

It is true that bituminous coals of the types listed above are generally accepted as the most desirable and easiest to handle in kiln firing. It is equally true that many others with different properties are being used satisfactorily. If color of the finished ware is not important, and often it is not, it would seem a waste of money to pay a high price for low-sulfur coal when a cheaper relatively high-sulfur coal is available. It is a fact that many heavy clay products plants habitually use high-sulfur coal for firing and, if the ware is properly dried, have little trouble with scumming. Coals with strong caking and coking qualities are also often used successfully, although additional work is necessary to keep the firebed open for the passage of air.

It seems likely that bituminous coal will continue to be the most important of the solid fuels for kiln firing but the producers should not deceive themselves by thinking that low-volatile coals cannot be used for this purpose.

Low-volatile Coals in Kilns

Low-volatile coals of the semibituminous and anthracite varieties are being used successfully for kiln firing. Quoting from J. T. Robson, ceramic engineer:[5]

> Very often where a straight nonflashed oxidized product is desired, a low volatile coal is and must be used for firing heavy clay products; also in the case of forced-draft kilns where exceptionally uniform results are obtained in the kiln and in the firing of glazed ware, a low volatile coal is desirable and exceptionally uniform firing is obtained.

H. C. Kenyon, superintendent of the Standard Brick and Supply Co., Charleston, W. Va., in discussing the changing of their kiln-firing fuels from natural gas to coal, says:[6]

> When we changed back to coal, we followed the book and purchased long flame coal but soon we were convinced that the place to burn coal was in the fire box and not in the kiln. We then tried semi-smokeless coal and the results were surprising.
>
> We are still using a semi-smokeless coal for all except the last 48 hours of the burn when at this point we change over to a low volatile smokeless coal which enables us to burn the ware in as clear bright colors as we obtained with gas firing.

Anthracite coal in the larger sizes was formerly used for kiln firing by brick manufacturers of Pennsylvania, New York and New Jersey. The increased value of this fuel for domestic use forced them to abandon it. Bituminous coal is generally used now.

H. G. Turner, State College, Pa., reports[7] that anthracite coal "is being used with marked economy over other fuels in a modern plant engaged in the manufacture of common and face brick of high quality," at Hazelton, Pa.

There seems to be a possibility that, with forced draft, the use of small-sized anthracite may prove profitable for firing periodic kilns in some plants in the eastern part of the United States. In an article in *Brick and Clay Record* (March, 1932) there is a tabulation that indicates that material savings might be made by substituting small-sized anthracite for bituminous coal in hand-fired periodic kilns at Allentown and Bloomsburg, Pa. The same article indicates the possibility of a material saving in fuel cost by substituting anthracite producer gas in a tunnel kiln for hand-fired bituminous coal in periodic kilns at Binghamton, N. Y.

In a number of plants manufacturing common brick, small-sized crushed anthracite is mixed with the clay in order to accelerate burning and produce a more uniform temperature throughout the kiln. This results in a decreased number of unsalable brick as well as producing a lighter weight brick. While some bituminous coal is used for this purpose, it is not as satisfactory as anthracite.

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[5] J. T. Robson: *Trans.* A. I. M. E. (1932) **101,** 254.

[6] *Brick and Clay Rec.* (March, 1932).

[7] *Trans.* A.I.M.E. (1932) **101,** 253.

The amount of coal so used is frequently from 50 to 60 lb. per 1000 bricks. In 1926 about 7,500,000,000 common bricks were manufactured in the United States. If crushed anthracite had been mixed with clay for all these bricks the coal so used would have amounted to about 200,000 tons. The actual market for this use is much less.

Except in a few localities, anthracite coal cannot be classed as a serious potential competitor of bituminous coal for kiln firing.

Stokers with Periodic Kilns

Much thought has been given to the problem of applying stokers to periodic kilns. Theoretically such equipment should reduce labor costs, save fuel and allow closer heat control. Considerable experimenting has been done by the Engineering Experiment Station of The Ohio State University and by ceramic manufacturers in attempting to make stoker firing of periodic kilns practicable. Most of the efforts have been devoted to developing and applying small stokers to each firing box on the kiln. The work of the Engineering Experiment Station was an effort to use one stoker per kiln. The results of a number of these attempts have been published by the Experiment Station and in various technical journals. Stoker firing has been controlled so as to produce ware of good quality. A few installations have been economically successful.

Continuous Kilns

Continuous tunnel kilns have been developed so that they are no longer an experiment. The point has been reached when we can almost say that the periodic kiln is obsolete in plants making tableware, artware, and floor and wall tile. Tunnel kilns are being used successfully for firing all kinds of heavy clay products except sewer pipe, flue liners and similar products. They have found great favor in the refractory industry. About 160 of these kilns are now in use in the United States. Some of the advantages of tunnel kilns are reduced labor costs, better heat control and consequently a better and more uniform product, and from 40 to 55 per cent saving in fuel. Some of the disadvantages are decreased flexibility in quantity of production and inability to flash where color control, other than clear straight colors, is necessary. These disadvantages are so great that there seems to be no danger that periodic kilns will soon be obsolete in the heavy clay products industry.

Stoker and hand firing of coal are both being successfully used with tunnel kilns. A few tunnel kilns are using powdered coal. Difficulties with fly-ash limit powdered-coal firing to the manufacture of low-grade products.

Producer gas can be used in either periodic or tunnel kilns. It has found greatest favor in the larger plants.

RIVAL FUELS

Although wood is a desirable fuel for kiln firing, scarcity and high cost have practically eliminated it.

Fuel oil has met with limited use in the eastern part of the United States but has found wider application in the Southwest and West.

Natural gas is a satisfactory and desirable fuel for kiln firing. It is used successfully in both periodic and tunnel kilns. Some of the advantages of natural gas are reduced labor costs, ease of heat control with accompanying improvement of quality of ware, practical elimination of smoke, and increased cleanliness of plant.

Natural gas has replaced much coal for the firing of tableware, artware, etc. It has also become a strong competitor of coal in the firing of glazed brick, refractory shapes, and high-grade refractories of all kinds. It seems doubtful whether coal can soon regain much of this market.

Great extension of gas pipe lines, and low priced natural gas for industrial purposes has led to the extensive displacement of coal by gas in ceramic plants during recent years, for the firing of all kinds of heavy clay products. This is particularly true west of the Mississippi River and generally in regions far from coal mines where cheap gas is available, but it is also going on in territory adjacent to coal mines. In other words, natural gas is a serious competitor of coal as a fuel for kiln firing. The problem is simply one of over-all economy. It is difficult to obtain reliable figures of comparative thermal efficiency of coal and natural gas as used for firing ceramic ware.

One Ohio brick manufacturer[8] is firing face and common brick with natural gas from the company's own wells. They report that the "gas consumed averages from 12,500 to 13,000 cu. ft. per 1000 brick. For the common brick, the consumption may be as low as 10,000 cu. ft. while for face brick it may be 15,000."

The average coal consumption to fire this quality of face brick in Ohio is 1320 lb. per 1000 brick. Assuming an average B.t.u. of 12,600 for the coal, we find that 13,000,000 B.t.u. from gas does the same work as 16,632,000 B.t.u. from coal.

The Hill Brick Co., East St. Louis, Ill., reports[9] a natural gas consumption of from 9,146 to 12,570 cu. ft. per 1000 brick. The B.t.u. value of the gas averages over 1000 per cubic foot.

The Cannelton (Ind.) Sewer Pipe Co. reports[10] the following test values of coal and natural gas in burning sewer pipe in periodic kilns:

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[8] *Brick and Clay Rec.* (Aug. 12, 1930) 210.

[9] *Brick and Clay Rec.* (Nov. 4, 1930) 531.

[10] *Brick and Clay Rec.* (Nov., 1931) 400.

	Fuel	B.t.u. per Ton Ware, Millions
Test No. 1	Natural gas,	9.496
24 in. Std.	Coal	10.07
Test No. 2	Natural gas,	8.57
24 in. D.S.	Coal	9.82
Test No. 3	Natural gas,	10.6
24 in. D.S.	Coal	11.1
Test No. 4	Natural gas,	9.18
24 in. D.S.	Coal	10.4
Test No. 5	Natural gas,	12.2
4 in.	Coal	14.5

Charles Butz, chief engineer of the Denver Fire Clay Co., reports[11] that they find 23,000 cu. ft. of 840 B.t.u. gas (19,320,000 B.t.u.) equal to one ton of 12,500 B.t.u. coal (25,000,000 B.t.u.).

These relative efficiencies of coal and natural gas are listed in the hope that they may be of value to coal men who are selling coal to the ceramic industry.

It might be of interest to know that increasing prices of gas in the eastern part of the United States have forced some companies to change from gas to coal.

MARKET POSSIBILITIES

It is estimated that the ceramic industry uses from 12,000,000 to 12,500,000 tons of coal per normal year. Part of this is used for power purposes, part for heating driers, but by far the greatest use is for kiln firing.

As previously indicated, tunnel kilns are rapidly replacing periodic kilns for firing the finer wares. Here gas is the generally used fuel. This market for coal, though once considerable, seems doomed to be largely lost.

It is in the firing of the heavy clay products that the largest coal market is found; that is, common brick, face brick, hollow brick, glazed brick, paving brick, hollow building tile, refractories, drain tile, sewer pipe, flue liners, wall coping, roofing tile, and various other vitrified bricks or blocks. (See Table 3.) Using the 1926 production and these figures

TABLE 3.—*Data from Brick and Clay Record* (*March* 26, 1929)

Product	Coal, Per Cent of Total Cost of Manufacture	Coal Used for Kiln Firing, Lb.
Drain tile	29	450 per ton
Common brick	28	965 per M
Paving brick	24	2520 per M
Face brick	22	1320 per M
Hollow brick	20	450 per ton
Sewer pipe	18	1100 per ton
Refractories		1675 per M

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[11] *Brick and Clay Rec.* (Dec. 31, 1929) 883.

of coal consumption, we find a potential coal market in the heavy clay products industry of the United States for kiln firing of 9,250,000 tons per year. Owing to displacement by other fuels, the actual market is somewhat less.

SELLING COAL TO CERAMIC PLANTS

No two clays can be fired in exactly the same way. Few clay plants have exactly the same firing schedule. No two coals can be fired in exactly the same way on any plant. Kiln firemen on each plant have a rather rigidly fixed routine for firing, developed to fit the needs of the clays and suitable for the coal being used. The routine used may not be the most saving of fuel but at least it does a reasonably satisfactory job.

If the coal salesman sells a coal that reacts somewhat differently from usual in the fireboxes, it means that, in some details, the firing routine must be changed. Failure to do this may ruin several kilns of ware. Firemen are human, they do not like change. They may not be competent to change quickly, consequently the new coal may be declared unsatisfactory when actually it may be a better and more suitable coal than that previously used.

Coal producers have recognized the necessity of fitting the coal to the power plant. Many employ experts to teach boiler firemen how to use their coal. Similar service rendered the ceramic industry in kiln firing should pay dividends. The natural gas industry has rendered such service.

Experiments have shown the possibility of affecting material fuel savings by better firing practices. Here is a fertile field for the coal salesman. He has an opportunity to increase sales by showing savings made possible with his coal. This should be one effective way to combat the competition from natural gas.

SUMMARY

1. High-volatile coals are the most desirable of the solid fuels for kiln firing, although low-volatile coals can be used.
2. Producer gas is a desirable fuel and profitable in some of the larger plants.
3. Natural gas is the greatest competitor of coal for kiln firing. It has largely replaced coal for firing the finer wares. It cannot compete with coal in firing heavy clay products except in regions where cheap gas is available.
4. Coal producers could increase sales by furnishing expert firing instruction.

DISCUSSION

(*Byron M. Bird presiding*)

W. E. RICE,* Pittsburgh, Pa. (written discussion†).—A study of the firing of kilns enables us to name the characteristics that are desirable in a coal that is to be burned in them. The desirable characteristics are those that will most easily produce the conditions that are wanted in the firing of a kiln. It does not follow that all who operate kilns can economically use a coal that has all these characteristics; nevertheless a knowledge of them can be helpful, for it indicates the precautions that must be taken to obtain the correct conditions with a coal that differs in some of the characteristics. It is true, as Professor Nold has stated, that many coals are successfully used that differ widely in their characteristics, and sometimes special conditions require different characteristics in the coal, but it is equally true that the process, the furnace used, or the method of firing must be adapted so that they will produce the correct results in spite of these differences. This is illustrated by the following examples:

1. If the ware is damaged by sulfur and if low-sulfur coal is not available at an economical price, the use of high-sulfur coal requires that the ware be thoroughly dried before it is placed in the kiln.

2. The burning of low-volatile fuels at a given rate requires the passage of more air through the fuel bed than the burning of high-volatile fuels, and this necessitates a greater difference of pressure to move the air through the fuel bed. When induced draft is used (most periodic kilns have only the induced draft from a stack), it is not desirable to increase the difference of pressure, because this is accompanied by reduced pressure in the kiln and hence inward leakage of cold air through the walls and uneven heating; therefore the furnaces are made large so that with a given difference of pressure the rate of burning per furnace is the same although the rate per square foot of grate is lower.

3. If small sizes, such as mixed nut and slack, are burned, the resistance of the fuel bed is high, and this difficulty is surmounted by supplying air to the ashpit under enough pressure to overcome the resistance of the fuel bed.

The furnaces of most kilns lack the refinements that provide accurate burning control, hence for best results they require coals that burn freely. It seems probable that the preference for "long-flame" coals in kilns is due to their burning freely, rather than to the length of flame.

G. A. BOLE,‡ Columbus, Ohio.—It is true that crude ceramic wares can be burned with high-sulfur coals. In fact, a great deal of ceramic ware of the cruder type is burned with high-sulfur coal. It is the local coal. A good part of the coal is mined with the clay from which the ware is made. It is of such quality as occurs there and no selection is possible.

The larger companies, of course, do buy their coal from the larger coal producers, who can give them better service. The large coal-producing companies could do the ceramic industry a great service by having an engineer familiar with ceramics, who could aid in the application of their particular fuels to the particular job. It is very largely that—the application of a particular fuel to a particular job.

The problem of long-flame coals versus short-flame coals is one that has been discussed in ceramic meetings for years. Theoretically there is an advantage in a long-flame coal, although many plants burn the short-flame coal successfully.

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* Associate Fuel Engineer, U. S. Bureau of Mines Experiment Station.

† Published by permission of the Director, U. S. Bureau of Mines.

‡ Research Professor, Department of Ceramic Engineering, Ohio State University.

Not only is there variation in coal; there is also a great variation in furnaces. We have what looks to a combustion engineer like an inefficient, foolish thing—a dead-bottom firebox. With the local coals, the higher clinker salts, this furnace is the best type of firebox for that particular set-up.

Two or three plants, where a smoke nuisance regulation was causing inconvenience, got along a little better by starting with a short-flame coal and later using a long-flame coal. Those plants happened to be burning refractories that required a higher temperature than is required in the ordinary plant.

I still think there is much hope in producer gas. Perhaps some will question this because their experience has been unhappy, owing, I think, almost entirely to their system of generation.

There has not been a small and efficient gas producer system on the market. The most successful application of producer gas to the ceramic industry is in what is called the chamber type of kiln—a continuous type of kiln where the sensible heat in the producer gas is not lost. It is possible that some smaller type of gas producer could be designed in which the sensible heat would not be lost and which could be applied in a system peculiarly suited to it.

Oil has been used successfully in the burning of ceramic wares. A system being tried now is, I think, going to make fuel oil much more available in the periodic kiln, even for burning the higher type of ware, than it has been in the past. Even glazed ware is burned in open kilns with this particular system of firing, and there is no trouble with fuel oil smoking during the early stages of the burn. Nor do they have the extremely hot, sharp flame that is normally developed in the later stages of the firing. It is a sort of surface combustion system. It is not completely worked out, but there are two or three special applications.

R. Purdy,* Columbus, Ohio.—My first experience, in 1899, was with gas at the Mosaic Tile Co. plant at Zanesville, which had just been opened. The gas company's burners were there and firing was done under the instruction of the gas company's engineer. The product being burned ranged from vitrified white tile to what they called clay colors. Speaking in terms of pyrometric cones, they were firing cone 10 in the top and cone 6 in the bottom of the kiln, a range of four times 36, or 144° F.

That was all right as long as the orders came for vitrified colors and clay colors in proportion to the variety of heats obtained in the kiln, but when the stocks piled up and the orders changed and we had to fire an entire kiln of vitrified colors or an entire kiln of clay colors, we were in trouble. Then the superintendent asked me to take charge of the firing. I worked with the kiln, staying up night after night as well as working during the daytime, trying to get distribution of heat. I failed. In desperation I went around town and talked to the kiln burners about it. I found one whose experience had taught him to take the burners out of the kiln and put in pipes instead. He told me to put in a one-inch pipe with a nozzle in the end, bore a hole for an aspirator, slip a pipe over that for a sleeve, knock a hole in the kiln above the hub, and give it a lot of auxiliary air. I did that and got even distribution of heat for the first time. I have been trying ever since to explain to myself why I got even distribution of heat in that great big kiln with that crude equipment.

Since then I have had experience with coal (anthracite and bituminous), gas and oil, and with every one of those fuels I have manipulated the kiln in the same sort of fashion in order to get even heat distribution.

My experience has been with kilns in which the ware had to have an even distribution of heat. We had to fire our kilns absolutely even from top to bottom, from side to center. Rarely do we find one plat of cones throughout the entire distribution of the kiln, top and bottom, center to side, from month to month, that differs from the others placed. That is how closely we have to fire grinding wheels in kilns. I have fired grinding wheels with anthracite with success, with soft coal with success, with gas with success, and with oil with success. In every case I have had to have auxiliary air. I have had to treat my firebox as a gas producer, not as a heat producer.

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* Secretary, American Ceramic Society.

Mr. Rice spoke about the rapid flow of air through the coal bed. Mr. Bole spoke of the dead bottom firebox. We had the same design in our anthracite kilns at Worcester, Mass. The grate bars were inclined to drop to the floor in the back. They were not grates, just pipes, just enough to support the bed. Rarely was there any heat or any light in that ashpit. We allowed the ashes to accumulate so that it was dead practically all the time, and the only time there was a light was after the ashes had clinkered. During the firing, we had to clinker occasionally, even firing with anthracite, but we kept that firebox dead. When the firebox showed any light, we placed iron aprons, made of sheet iron, against the kiln so as to prevent air from going through the coal bed, except such as was necessary to support the combustion, the natural partial combustion.

During the war embargoes on freight cars from Pennsylvania, we were faced with the necessity of burning soft coal from West Virginia. We changed our procedure as to manipulation of the dampers and of the firedoor and of the operation above the hub of the kiln that lead directly into the bag wall, which is the chimney in the kiln from the firebox. We had to change our instructions and our regulations in order to handle this soft coal, but our procedure with the soft coal was the same as with the hard coal.

Recently a gas company put gas in a kiln at Dayton, with a pre-mix and a lot of expensive gadgets. The purpose was to demonstrate that the kilns could be fired evenly and with small cost, with no dirt and no ashes. The gas company was allowed to fix up a 5-ft. kiln. Not once was that kiln fired with even distribution of heat; not once was it successfully fired, until finally the company turned the problem over to me. I bypassed the pre-mixer, took out the gadgets, put in my one-inch pipe, broke holes in the firebox, broke a hole in the side of the kiln above the hub, letting air into the bag wall, and proceeded to fire with oxidizing conditions, with absolutely even distribution of heat.

That may seem radical. These are all kilns inside buildings and fire wares that require even distribution of heat, not brick kilns out in the open that can take a wide variation in heat. Even the most even heat that they experience is probably 100° apart from the top to the bottom. The kilns with which I have had experience required absolutely uniform heat distribution. We kept our grates cluttered up with ashes. We fired with our ashpit partly closed, practically closed at times. We brought in our auxiliary air over the fire and above. Our scheme was to get the firebox to produce gas, which went into the kiln and, by contact with the air, the two streams of gases would mix, combust, and give an even distribution.

The paper says that the combustion is seldom seen except a few feet from the firebox. Gas analysis shows us more oxygen in our outgoing flues than we can account for in any other way, excepting air; we had to argue that we had streams of air and streams of gas. We know that streams of gas do not commingle readily. They have to be run into a baffle before they commingle. In other words, a kiln cannot be fired on the same basis as you would fire boilers or fire for efficiency in a firebox. We are not after firebox efficiency; we are after kiln efficiency, or 100 per cent distribution of heat. The firebox must be used more as a gas producer than as a heat producer. This has been my experience in the past 30 years.

The Carborundum company had horizontal grates and little secondary air. The fire bed was so thin that the heat from the furnace varied. It developed what in the pyrometric cone is known as "freezing." The company discovered that these cones could be made to set in whatever position they were chilled in, and it would require a heat treatment far greater than was necessary to bring the cones down if they were chilled. I never heard of a chilling or freezing of a pyrometric cone except in kilns with a thin fire bed, where an effort is made to combust all the fuel within a few feet of the firebox or grate, just as Professor Nold described. We have never had any of that trouble where we fired as I have described. The same was true of oil.

Solid particles *are* obtained from the firebox. A lot of solid particles do go through the kiln. But to believe that all transference of heat is due to these particles rising from the firebox and going through seems ridiculous. I do not know why you cannot take into account a considerable amount of gas that is partly combusted going through with air and being fully combusted in the kiln. I do not deny these incandescent particles. They are there. But I am hoping that some day fuel engineers will acknowledge that there is uncombusted gas going through the kiln and that that is the only way to get even heat distribution.

R. Purdy (written discussion).—This paper is excellent in its content and construction, and I am sorry that I have to differ with any statements therein, but I certainly do not agree with the first paragraph on page 381 which minimizes the incomplete combustion of gases in the kiln.

Gas samples cannot be taken any place in the kiln or kiln flues without finding a considerable amount of CO and a considerable amount of free ozygen. The flame seen in the outlet flue is certainly a real flame caused by combustion of gases.

Kiln operators cannot get away from the fact that the firebox is a gas producer, and not a heat producer, and any failure to recognize this will result in failure.

Stokers have failed as firing media for periodic kilns because it has not been recognized that the entire combustion is not in the firebox if an even distribution of heat is to be obtained.

The less volatile material there is in the coal, the more dead bottom must be the management of the firebox to obtain the maximum heat distribution with the minimum fuel consumption, and this is true whether the firebox is really dead bottom or has grates. During the war, when it was difficult to get coal in New England, I was asked if it were possible to fire bituminous coal in the periodic kilns of the Norton Company obtaining even distribution and not causing a smudge in the factory. My reply was "Yes," but that it would require that more attention be given to the clinkering, and that a different adjustment would be required of the auxiliary air inlets and ashpit coverage. We were very successful in firing bituminous coal, although the fireman had had experience with only anthracite coal.

In Dayton, Ohio, I succeeded in firing the kilns without smoke by covering the ashpit with steel hoods, allowing the ash and clinkers to accumulate, preventing any amount of air going through the grates, and obtaining all of the air necessary for combustion over the fire, through the fire door, and through a hole in the kiln wall just above the hub leading into the fire bags. The fresh air thus admitted above the fire kept the top of the kiln cool, and caused the combustion to take place throughout the kiln chamber.

Bole and Rice in their report on problems of firing refractories, in U. S. Bureau of Mines *Bulletin* 271, were absolutely wrong when they said that combustion under normal conditions is completed within a few feet of the fuel bed in kilns that have the maximum heat distribution and minimum fuel consumption.

The failure to realize that fireboxes in kilns should be gas producers, rather than heat producers, by the firemen who in the early days were trained by the Carborundum company brought to light the problem of the freezing of pyrometric cones. The use of light fire beds and the disuse of auxiliary air supply coming in above the fire bed rather than through it caused temperature fluctuations that were reflected in the cones. I have had experience with the old Carborundum burners, and the operators were shocked when I insisted that they use heavier fire beds, clean less often, cover the ashpit with hoods, keep the fire doors partly open, and make an opening through the kiln wall just above the hub leading into the fire bags. I succeeded in convincing them, however, that by employing my method they could fire the kiln more uniformly oxidizing, with better heat distribution, less fuel and without freezing of the cones.

W. A. Koehler,* Morgantown, W. Va. (written discussion).—One objection to sulfur in coal for firing ceramic wares is that in a kiln there are almost ideal conditions for the manufacture of sulfuric acid; namely, sulfur dioxide, moisture and oxidizing conditions, with the clay serving as a catalyst. An average periodic kiln will hold 100,000 face bricks. According to Table 3, face bricks require about 1320 lb. coal per 1000 bricks, which corresponds to 132,000 lb. coal per kiln run. Rarely do we find a coal used for firing brick that contains less than 1 per cent sulfur.

Sulfur in the coal for firing a kiln of bricks amounts to 1320 lb. if the coal contains but 1 per cent of sulfur. Not all the sulfur in the coal leaves the stack in an oxidized form, but the sulfur retained in the ash rarely amounts to 10 per cent of the total sulfur. But even on this basis, the sulfur in the coal available for sulfuric acid manufacture amounts to 1188 lb. per kiln run. This corresponds to 3600 lb. H_2SO_4, and it is probable in many cases that the sulfur does form this amount of acid or its equivalent in SO_3, which eventually forms H_2SO_4. In terms of concentrated commercial acid this amounts to 230 gal. Assuming that the kiln is on fire for 6 days, the kiln produces the equivalent of 600 lb. H_2SO_4, which corresponds to 40 gal. commercial acid per day.

It is generally believed that sulfuric acid, SO_3 or SO_2 will not affect iron or steel as long as they are in the gaseous state. In a periodic brick kiln there seldom is any iron in contact with the kiln gases. In car tunnel kilns, however, it is quite common to pass the products of combustion through sections of iron or steel pipe; in some cases the stacks are also of steel. The gases usually are cooled to the dew point before they leave the stack. Therefore, as the stack temperature drops, the sulfur trioxide unites with the water vapor, forming sulfuric acid vapor. When the temperature has dropped to the boiling point of concentrated sulfuric acid, about 620° F. (327° C.) the sulfur trioxide has practically all united to form sulfuric acid, and the acid condenses as a 98.3 per cent acid, which becomes more dilute on cooling owing to the absorption of water. In steel stacks the corrosive action[12] of the acid can readily be observed. This can be observed in the steel stacks of tunnel kilns burning natural gas, in which the sulfur content is exceptionally low.

A few years ago I made an inquiry of the potteries in West Virginia, and one of the questions asked was: Irrespective of cost, what do you consider the ideal fuel for firing whiteware? Whiteware is among the higher classes of ceramic products. Several of the manufacturers said that they had often contemplated changing from gas to coal for economic reasons, but one potter mentioned that, irrespective of cost, he preferred coal for his fuel. He added that of course it must be the right kind of coal. Perhaps Mr. Purdy's statement about the uniform heat distribution is an important factor in this. I think it is of particular interest that one potter so expressed himself, and probably others have the same notion, that coal is the ideal fuel for firing the high-grade wares, not merely the heavy clay products but white wares.

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* Department of Chemical Engineering, West Virginia University.

[12] J. F. Barkley: The Sulphur Problem in Burning Coal. U. S. Bur. Mines *Tech. Paper* 436 (1928).

H. E. Nold.—Mr. Purdy has opened up the age-old argument in the ceramic industry as to how a coal is burned and how the gases from coal are burned, and how the heat gets to the ware. Frankly, I am not a ceramic man and do not claim to be an expert on combustion in ceramic kilns. However, my quotations and citations were made from *Bulletin* 271 of the U. S. Bureau of Mines, where several pages of gas analyses are given, which were made especially to test out this proposition of where the combustion takes place and how it takes place. Firing under different conditions, they found exactly what would be expected. Above the fire, with normal firing, they found a considerable amount of carbon monoxide, hydrogen and methane. The auxiliary air, coming in over the top, starts to burn the gases and completes their combustion within a few feet. Before the gases go down into the ware, the carbon monoxide, hydrogen and methane normally disappear. In a few cases they choke-fired and then there was not enough secondary air, and the carbon monoxide carried down through the ware. I assume that the U. S. Bureau of Mines men knew how to set up the tests and knew how to take gas samples.

Regardless of how and where the combustion takes place, it is a fact that practically any type of fuel can be used satisfactorily to burn ceramic ware, and that the fires must be handled differently with the different types of wares and fuels.

I still believe that you people who are producing coal within reasonable distances of ceramic plants have been overlooking a considerable market.

R. H. Morris,* Ansted, W. Va. (written discussion).—Fuel used in the ceramic industry, whether it is coal or a substitute, is, it would seem, primarily for the purpose of furnishing heat for driving off the moisture from the different kinds of formed wares—also for the purpose of firing (vitrifying) these wares. Generally speaking, coal should be the cheapest fuel when compared on a B.t.u. basis with any of its substitutes; therefore, if coal can be made to serve the purpose at a comparable cost of installation and operation, it will be used. The substitute fuels no doubt have an advantage in that there is no residue to be handled after combustion has been completed but modern methods are available to handle the residue from coal automatically.

Mr. Kenyon is referred to as having stated that the place to burn the coal is in the firebox and not in the kiln, which eliminates the necessity of a long-flame coal; this being the case, there would seem to be two methods possible of producing heat to be conducted to the kiln. The first method is by stoker, which has been mentioned as being used. The stoker manufacturers no doubt can supply or develop a stoker with automatic control and also with ash-removing equipment, as such equipment is now on the market. Second, the use of coal that has been made into coke would also seem to offer an ideal fuel for producing heat to be conducted to the kilns. The author has brought out two very important points in the utilization of coal:

1. There is shown indirectly the great need of more research work in coal, which is not getting the attention it should have.
2. The matter of servicing coal-consuming industries, such as the ceramic industry, is important and, as noted by the author, fuels competing with coal pay better attention to service. The matter of service is important even after the best equipment and proper kind of coal or coke are secured.

The ceramic industry, as cited by the author, offers a large market for coal and certainly only awaits the proper effort to secure this tonnage.

D. B. Hendryx,† Pittsburgh, Pa. (written discussion).—No attempt is made in the following paragraphs to discuss Professor Nold's paper adequately. A few observations from experience in firing heavy clay products are made, and some additional data are presented concerning relative efficiencies of coal and natural gas.

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* General Manager, Gauley Mountain Coal Co.

† Chief Engineer, Harbison Walker Refractories Co.

The problem encountered in the firing of any heavy clay product in a periodic kiln is that of carrying the heat to the center and bottom of the kiln without overheating the ware nearest to the fireboxes. The fireman always concentrates his entire attention on the center of the kiln. His only interest in the front heads is to see that they are not heated too rapidly and do not reach too high a temperature.

Ceramic materials are relatively poor conductors of heat. The temperature of the entire body of any single brick or refractory shape must be raised as evenly as possible. If an outer surface or a thin portion is heated to a higher temperature faster than the body of the shape, it will shrink (or expand) more rapidly than the center, resulting in cracking and distortion. The larger the shape, the greater the difficulty. Therefore the gases that carry the heat to the brick can be maintained at a temperature only a little higher than that of the ware.

Bricks are set in a kiln in large masses of checkerwork, with only very small spaces between the bricks through which the gases can pass. The bricks with which the gases first come into contact soon approximate the temperature of the gases, after which very little heat is absorbed by these bricks and the hotter gases penetrate farther into the setting. The gases give up heat to the bricks with which they first come into contact, until the ware very nearly reaches the temperature of the gases. The gases cannot then give up heat to the outer bricks, so they carry it deeper into the setting before releasing it to raise the temperature of cooler ware.

The amount of heat released by the combustion of one pound of coal, if transmitted entirely to the products of combustion, could be carried by about 24 lb. of gases at 2000° F., or by about 16 lb. of gases at 3000° F. The greater amount of gases would carry the heat farther into the setting and distribute it more evenly to the ware without the danger of local overheating.

All of which goes to explain the reason why so many combustion engineers, representing coal and gas companies or hired by ceramic manufacturers, often make dismal failures of their first attempts to reduce fuel consumption. They are amazed at the amount of excess air used by the regular plant firemen, particularly when kiln temperatures are below 2000°, when as a matter of fact this air is necessary to "dilute" the flame temperature to the range where it can safely be allowed to come into contact with the ware.

When the amount of combustion air is reduced to a point approaching theoretical proportions, the top of the kiln is soon overfired, while the bottom may scarcely reach a red heat. Boiler practice, where cold tubes of high conductivity absorb much heat from high-temperature flames and fuel beds, is hardly comparable in any way with kiln-firing requirements.

The heat of the fuel can be released entirely in the fireboxes, and the temperature of the gases reduced when necessary by the admission of sufficient excess air. This is usually done when burning clear colors in building brick.

When firing refractory products and other ware where "flashed" colors are not objectionable, the heat may not all be released at once. Combustion may be delayed so that part of the latent heat is carried past the nearest bricks in the form of undeveloped chemical energy, and released by combustion around the colder ware. It is fairly easy to do this, at least intermittently, when using a high-volatile coal, which accounts for its deserved popularity in many ceramic plants. High-volatile coal possesses the undesirable feature that there is danger of losing much of the most valuable part of the fuel unless it is fired in a type of furnace that distills the gases slowly, or unless furnaces are fired more lightly and frequently than is usually the practice.

It is more difficult to delay combustion when using low-volatile coal, although it can be done by using a thicker fuel bed, operating each firebox as a small gas producer, and admitting secondary air in a manner to prevent too rapid mixing with the combustible gases. Delayed combustion can be used with natural gas, and good flashed colors obtained.

A fireman experiences his greatest difficulties with coals that clinker badly at the temperatures required to fire the ware. A coal that would be satisfactory for burning common red brick might not have an ash with high enough fusion point for firing high-temperature refractory kilns. Firemen also object to coals that are "sticky" when first fired, and to those that cake together badly, making it hard to maintain an even flow of air through the fuel bed.

TABLE 4.—*Comparison of Fuel Consumptions*

Fuel	Kind of Brick	Type of Kiln	Maximum Kiln Temperature, Deg. F.	Firing Time, Days	B.t.u. Value of Fuel Used[a]	Millions B.t.u. per Ton of Ware Set	Ratio of Natural Gas to Other Fuel on B.t.u. Basis, Per Cent
Natural gas	Red shale face	Rectangular down-draft	1900	5–6	1,000	4.38	73.8
Coal, 25 per cent volatile			1900	5–6	12,500	5.93	
Natural gas	Machine-made refractory	Round down-draft	2300	10	1,050	5.8	74.4
Coal, 34 per cent volatile			2300	10	13,000	7.8	
Natural gas	Refractory sizes and shapes	Round down-draft	2200	8	1,050	4.66	75.5
Coal, 27 per cent volatile			2200	8	13,000	6.18	
Natural gas	Machine-made refractory	Tunnel	2400		1,000	2.8	
Coal, stoker fired		Tunnel	2400		12,500	4.93	56.8
Coal, producer gas		Tunnel	2400		12,500	4.15	67.6
Natural gas	Refractory shapes	Round down-draft	2300	9	950	7.91	88.
Coal			2200	9	12,000	9.02	
Natural gas	Refractory sizes and shapes	Rectangular down-draft	2400	9	950	7.55	96.
Coal			2300	9	12,000	7.83	

[a] Natural gas, per cubic foot; coal, per pound.

In tunnel kilns it is difficult to fire coal directly and to obtain combustion of the gases around the ware on the cars without exposing the ware to direct radiation from the hot fuel bed. When using natural gas, producer gas, or oil, the operator has better control of the velocity and character of the flame, and excellent results can be obtained.

Table 4 gives some comparisons of coal and natural gas on various types of kilns and ware, fired at different temperatures. In the last two cases given, gas-fired kilns reached higher temperatures and were set with higher percentage of heavy shapes than the coal-fired kilns.

C. E. BALES,* Ironton, Ohio (written discussion).—The Ironton Fire Brick Co. is now successfully using byproduct coke-oven gas for firing high-grade refractories. So far as I have been able to learn, we are pioneers in using this fuel for this purpose.

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* Vice President, Ironton Fire Brick Co.

J. L. CARRUTHERS,* Columbus, Ohio (written discussion).—As Professor Nold says, the firing of kilns must be considered from the standpoint of over-all economy, which takes into consideration time, labor, maintenance, fuel cost and ash disposal, as well as percentage of A-1 ware produced. Often the item of cleaning and repairing fireboxes, made necessary by excess clinker and slag, is a serious one, bringing about increased labor costs and labor turnover, and low kiln turnover and expensive repair charges. Therefore the fusion point of the ash and the cleanliness of the coal are important.

The suggestion that trained engineers be used to service the sale of coal to ceramic plants is good, but it should be realized that firing of kilns varies in many respects from practices followed in firing boilers. Often a high efficiency in the combustion process is not desired, since the color effects to be produced may require a high percentage of excess air or a deficiency in the air supply. The engineer must not only be trained for combustion work but must have a knowledge of the ceramic technology involved in the firing process. In addition, a knowledge of kiln design, operation, ware setting, and choice of refractories will be required if the engineer is to produce satisfactory results. The area in which sales engineers could operate is large, because unfortunately many large coal-consuming plants cannot employ an engineer for this type of work. Many factories are in need of this help. While the coal seller ordinarily should not be expected to furnish it, if he wishes to obtain new markets and maintain present ones, I am sure that the rendering of this service will be a profitable investment.

J. A. TEEGARDIN,† Columbus, Ohio (written discussion).—If you will trace on the map the clay industry in the state of Ohio, you will find that where there is a clay plant there is also found close by coal with which to burn the clay. There are exceptions to this, but it is almost universally true as far as the heavy clay products and refractories are concerned. Often the raw material and the coal are being mined on the same property. In other places, as this coal supply has become exhausted the clay manufacturer has been forced to seek a new supply. This he has been able to do without extra cost to himself because of the advent of good roads and fast trucking. In many instances he is saving money because he has been able to do away with railroad tracks, trestles and other unloading equipment. He can get a steady supply delivered to his kilns without extra handling and is not obliged to have his money tied up in large stocks of coal, as some manufacturers are forced to do in shipping by rail. For the most part, run of mine coal is used, often of very ordinary quality. The kiln burner at these plants, through years of experience, has learned to use this type of coal and, as Professor Nold has pointed out, does not like to change. However, the clay products manufacturer not only is interested in saving money on his fuel bill, but is more interested in getting a high percentage of first-grade ware. He can well afford to burn a few more pounds of coal if it will give him more money for his product in a higher percentage of firsts.

There is now a more awakened clay industry, especially in the heavy clay field. This is caused by the proved practicability and merit of new machinery placed on the market in the last year or so, which makes it possible to use vastly inferior clays and shales in the manufacture of these products. Formerly this was impossible. To me this means that ceramic plants will spring up closer to the centers of population, doing away with the expensive freight charge on the finished product, but requiring the use of either coal, gas or oil for burning. This will mean an added market for the producer that ships by rail. The clay manufacturer is also watching closely the new ideas in building homes and other structures. These things, even though business during these days is almost at a standstill, are causing the operator to survey his plants and products and he is looking for means of saving money and improving his output.

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* Associate Professor Ceramic Engineering, Ohio State University.

† Vice President, New York Coal Co.

It seems to me that the coal producer has a big opportunity to help the clay manufacturer to recovery and also to help himself. The ceramics industry has been investigating stokers, gas producers, and various types of kilns, hoping to find some way to lower fuel cost, which next to labor is the greatest item of cost. Professor Nold has pointed out these items but there is still much work to be done and many data to be obtained. In the next decade we shall see a greater use of the continuous car kiln. We think this is one place where stokers can be applied, which will mean a use for coal and a saving in money to the clay manufacturer. The use of stokers in the ordinary round or rectangular periodic kiln, we believe, is of doubtful value to the clay industry as a whole, owing to the amount of machinery and equipment, and investment required. But in the continuous kiln, where the stokers are being used continuously, we believe it will show a decided improvement. This should be investigated. At the plants now in operation, the coal industry can and should increase the sale of its coal, not only through expert firing instruction but also by improving the design of kilns, such as fireboxes, kiln bottoms and proper size of stacks.

The application of fuel for the production of ceramic ware is important. It should be based not on the first cost of the fuel, but rather on the composite of the different cost items that go to make up the total cost per unit of ware. The percentage obtained of ware of first quality is a very important item in operation of ceramic plants.

The adaptability of the fuel to the operating conditions of the individual plant plays an important part in the cost problem. When considering coal, three adaptability items might be set up: (1) burning characteristics, (2) coal size, (3) sulfur content in coal.

The burning characteristics and size of coal may be considered together because a change in either may accomplish an improvement in the adaptability of the fuel in a particular plant. If, by the use of a certain coal, the temperature in the kiln is maintained more satisfactorily, the plant operator might well expect less warping, cracking and blistering, and an improvement in the color and glaze of the ware, so the result will be an increase in the percentage of first quality ware. Better temperature control in the kiln frequently makes it possible to increase the temperature more rapidly and thereby reduce the total burning time of the kiln cycle. The capacity of the plant may thus be increased with no increase in firing labor cost because of the shortened cycle, and at the same time the percentage of first quality ware may be increased. It is not difficult to see how enormous savings may be accomplished in a ceramic plant if a coal is used that will reduce the length of burning time and also increase the percentage of salable ware.

The sulfur content of the coal often plays an important part in the percentage of first quality ware obtained. Sulfur has a great affinity for the material used in many ceramic plants and may cause blistering, checking and unsatisfactory glazing of the ware. In such plants coal with a low sulfur content is essential to satisfactory production.

Gas is the greatest competitor of coal in the ceramic industry. In these times it is especially so, when the gas producer has an excess of gas to sell. We are informed that in Huntington, W. Va., the Chamber of Commerce is offering gas at 15¢ per 1000 cu. ft. as an inducement for plants to locate there. In Ohio gas for industrial use is sold to the large consumer from 27¢ per 1000 cu. ft. up to 35¢ per 1000 cu. ft. In the heavy clay products field I do not believe we have anything to fear from gas at these prices, but gas at 15¢ per 1000 cu. ft. is another thing. At this price there are many manufacturers of clay products who would change at once to this fuel if they were given a long-time contract and guarantee of supply.

There is no question that we have lost out in the ceramic industry to other fuels, especially for the finer wares. Again it is not the saving on fuel so much as it is getting a better product and more first quality ware. Professor Nold has stated that natural gas has replaced much coal for the firing of table ware, artware, and is becoming a strong competitor of coal in the firing of glazed brick, refractory shapes and high-grade refractories of all kinds. He further states that it is doubtful whether we can regain this market. At this time ceramic manufacturers are trying to find other products to make with their present equipment. Those who have changed go into the finer products where cleaner fuel is necessary, and several plants that have used coal as a fuel have changed to gas or oil.

It seems to me that our problem is to investigate and ascertain whether our product is suitable for burning these finer ceramic wares in the kilns now used, and if not to try and develop kilns and equipment in which our fuel will produce as high-grade a product as our competing fuels.

E. LOVEJOY,* Columbus, Ohio.—On page 381, the long flame extending through the ware and at times appearing as stack "tassels" is attributed to incandescent particles of ash and carbon. The likelihood of delayed combustion being the cause of the flame is questioned. The denial of delayed combustion is clinched by the statement that "combustion . . . under normal conditions is completed within a few feet of the fuel bed."

In normal firing, immediately after baiting the fires we get a dense smoke through which the kiln flame and stack "tassels" are but dimly seen. With the disappearance of the smoke we get a clear view of the flame which during 10 to 15 min. gradually dies away and we have a clear kiln, without flame or smoke, until the next firing.

I was under the impression that smoke was rather difficult to burn. At any rate it comes from an incandescent furnace, accompanied by excess air, passes among a mass of closely set, highly heated ware, thence through hot floor and flues to and through the stack as smoke. Evidently it is not highly inflammable and though there is a flame seen dimly within it I cannot believe the flame is from combustion of the smoke. The flame continues for a short period after the smoke is largely eliminated.

Ash is set free during the combustion of the fixed carbon. There would be no ash, or a minimum, during the volatile gas stage of the furnace operation, since the fresh fuel, itself at first not ash-producing, covers the burning ash-producing coke. We have therefore a maximum long flame in the kiln when the fly-ash is lowest in amount and no flame when the ash is highest.

In forced-draft firing we get excessive fly-ash throughout the firing and especially after the volatile gas is driven off, yet the luminous flame comes and goes as in natural draft firing and is no greater. In one stoker operation that I examined there was, of course, excessive fly-ash but no luminous flame. The fly-ash does not seem to fit into the luminous flame.

Particles of burning fuel could be carried along in the gas stream and would develop a flame. To assert that such floating particles are limited to the volatile gas stage in order to correlate them with the luminous flame is going a bit too far. It seems to me more likely that such light particles will come off with the ash from the subsequent burning of coke.

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* Consulting Engineer.

R. Purdy.—May I again draw attention to the fact that the only kilns with which Messrs. Bole and Rice have personal experience are brick kilns in which even heat distribution is not deemed necessary. They have had no experience with the many kilns firing ceramic wares and in which even heat distribution is imperative. Therefore they are not qualified to make sweeping statements relative to "ceramic kilns." It is questionable whether Professor Nold is entitled to compare data from a half-dozen brick kilns with the many other ceramic-kiln operations where exactness in firing is imperative.

Furthermore, whereas in the brief and limited experience of Messrs. Bole and Rice the sulfur content of the coal was not a matter of consideration, it is a fact that in most ceramic firing sulfur is disastrous. It is wrong to conclude "that any old coal can be used in ceramic kilns."

H. E. Nold (written discussion).—The author's explanation of normal combustion in a kiln has drawn rather sharp criticism from Messrs. Purdy and Lovejoy. The author based his statements on the results of a rather extensive series of tests on Combustion in Kilns Burning Refractory Ware recorded in U. S. Bureau of Mines *Bulletin* 271, on pages 150–161. This does not seem the proper place to open a lengthy discussion of combustion in ceramic kilns. A careful study of the pages above cited should convince anyone that the statements regarding normal combustion in ceramic kilns in this paper are substantially correct.

Report of Committee on Methods of Valuing Coal Properties

INTRODUCTION

IN September, 1930, H. N. Eavenson, then Chairman of the Coal Division, appointed a Committee on Methods of Valuing Coal Properties, with John B. Dilworth as chairman. The latter prepared a schedule[1] of features for consideration under the general headings of Undeveloped and Developed Coal Properties. These subjects were divided among the members of the committee and during the next year written discussions of his allotted subjects were received from each member. Mr. Grimes and Mr. Eavenson discussed the entire list. These discussions were mimeographed and a set sent to each member before the meeting of the Institute in February, 1932. There was a general discussion at that meeting. At the meeting in February, 1934, Mr. Dilworth submitted a report,[2] which summarized the accumulated opinions and data and presented the conclusions of the committee as a whole on the subjects embraced in its schedule. Members of the committee expressed the desire that some of the individual opinions be published for reference. Mr. R. Dawson Hall's abstracts of these papers are therefore presented herewith.[3]

PERSONNEL OF COMMITTEE

JOHN B. DILWORTH (Chairman), Consulting Mining Engineer, Edward V. d'Invilliers Engineering Co., Philadelphia, Pa.

ELI T. CONNER, Consulting Mining Engineer, Scranton, Pa.

W. G. DUNCAN, JR., Superintendent, W. G. Duncan Coal Co., Greenville, Ky.

HOWARD N. EAVENSON, Consulting Mining Engineer; Eavenson, Alford & Hicks. Pittsburgh, Pa.

GEORGE WATKIN EVANS, Consulting Mining Engineer, Seattle, Wash.

J. A. GARCIA, Consulting and Constructing Engineer, Allen & Garcia Co., Chicago, Ill.

JOHN ALDEN GRIMES, Valuation Engineer, Bureau of Internal Revenue, Washington, D. C.

OTTO HERRES, Assistant Manager, United States Fuel Co., Salt Lake City, Utah.

J. R. HICKS, Consulting Mining Engineer, Eavenson, Alford & Hicks, Pittsburgh, Pa.

ROBERT D. REEDER, Mining Engineer, U. S. Bureau of Mines, Washington, D. C.

SAMUEL A. TAYLOR, Consulting Engineer, Pittsburgh, Pa.

BLEECKER L. WHEELER, Mining Engineer, Ford, Bacon & Davis, New York, N. Y.

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[1] See page 401. [2] See page 402. [3] See page 412 et seq.

SUGGESTED FEATURES FOR CONSIDERATION

A. Undeveloped Coal Lands

1. Should basis of coal valuation be: (*a*) Actual sales prices of similar coal? (*b*) Royalty received for similar coal? (*c*) Estimated earnings from operation?
2. Should basis of surface valuation be:
   (*a*) Current price of similar farm or uncultivated land, regardless of area owned?
   (*b*) Special worth of particular surface required for mining development?
3. How allow for fact that two otherwise similar coal tracts may be so located that one will cost more to develop than the other?
4. How allow for insufficient prospecting or uncertainty as to quantity or quality of available coal?

B. Developed Coal Property

1. Should coal and surface be valued separately from plant and mine, or
2. Should entire property be considered as an industrial unit and valued as a whole? If so, what is proper method of procedure?
3. If plant and mine be valued separately from coal land, should basis be
   (*a*) Cost to reproduce new, less depreciation for age, condition and obsolescence?
   (*b*) Original cost, less depreciation?
4. When mining town is so located as to have earning power independently of mine (houses rentable to non-employees), should it be valued separately from rest of plant?
5. Establish proper depreciation rates for the various classes of mine buildings, machinery and other equipment.
6. What ratio does salvage value of mine plant bear to its cost new?
7. How should the normal profitable life of a coal mine be determined?
8. Where property has more coal than will be exhausted by the existing mine in its normal life, how should excess be valued?
9. Where the coal in a mining area has not been thoroughly proven, should its tonnage be subdivided into some such classes as "proven," "probable" and "possible" coal and appraised accordingly?
10. What should "Mine Development" include, and how should its value be determined in the case of
    (*a*) A new mine not up to designed output? (*b*) An old mine nearing exhaustion?
11. If valuation be based on capitalized earnings or royalties, what rate or rates of interest should apply?
12. If above appraisal method be used, should income and life be based on actual or potential rate of production?
13. Is it possible for a given property (developed or undeveloped), to have two or more values simultaneously? For example: (*a*) Purchase-and-sale value. (*b*) Assessment value for taxation purposes. (*c*) Bankers' value for financing purposes.
14. Does value of property vary with operating and marketing ability of operator?
15. In case of leaseholds, how determine: (*a*) Value to owner or lessor? (*b*) Value to lessee?
16. What percentage of recovery should be used where: (*a*) Pillars are drawn? (*b*) Pillars are not drawn?

Report of Chairman of Committee on Methods of Valuing Coal Properties

By John B. Dilworth

The Chairman of this Committee presents herewith a digest of the opinions of its members on various phases of the subject submitted to them for consideration. Most of these opinions were presented in writing by one or more members. Each of them was submitted to every member of the committee for review, after which they were discussed in a general meeting. Likewise this digest has been submitted to, and approved by, every committee member, so that the opinions recorded here represent the views of the committee as a whole.

For convenience of treatment, coal properties have been divided into two classes: Developed and Undeveloped. Developed properties are those with mines producing or ready to produce coal. Undeveloped properties are coal lands not tributary to any existing mine.

Developed Properties

It is the consensus of opinion of this Committee that the true value of a developed coal property is the present worth of the prospective net earnings of that property. This fundamental principle has been asserted by numerous authorities on the valuation of mining properties, among whom may be mentioned the A. I. M. E. Committee on Federal Taxation, the Engineers Advisory Committee of the U. S. Coal Commission, Herbert C. Hoover, James R. Finlay, Walter R. Ingalls and Henry Louis.

The essential factors involved in a determination of value by this method are:

(*a*) Net earnings of the property per period.
(*b*) Life of operation.
(*c*) Interest rate or rates.
(*d*) Salvage value at end of life.

Each of these fundamental factors involves numerous elements, which must be either determined or assumed for the particular property under consideration. It is not the purpose of this Committee to consider all of these elements, but rather to offer opinions on the more important ones in the handling of which there seems to be no generally accepted practice among appraisers.

Assuming that a given property has but one true value at a certain time and that the measure of that value is the present worth of its future earnings, it follows from this premise that no independent value attaches to the essential components of the property. In actual practice, however, it is usually convenient and desirable to assign certain portions of the total valuation to such major subdivisions of the property as plant, town, surface land, mine development and coal reserves.

Mine Plant

In regard to mine-plant valuation, the several committee members who express themselves on this subject generally agree that the present value of a mine plant is the cost of reproduction new at current prices of equipment, material and labor, less depreciation for age, condition, obsolescence, and period of usefulness. This value, of course, must not exceed the total value of the property as determined from prospective earnings.

Mr. Grimes notes that plant valuations for Federal income tax purposes must be based on original cost (if assembled since March 1, 1913) less depreciation. He also favors this basis when the appraisal is to determine the amount of depreciation to be charged against operating costs. In connection with charges for plant obsolescence he remarks that the appraiser should take account of the possibility of replacing the plant by a more modern one, which would reduce operating costs and thereby enhance the value of the entire property.

Plant Depreciation

The general question of mine-plant depreciation has been given careful study by Mr. Duncan, who has compiled a comprehensive and valuable table of depreciation rates for mine-plant structures and equipment, based mainly upon publications of the U. S. Bureau of Internal Revenue and of A. W. Hesse. A brief summary shows:

CLASSES OF ITEMS	YEARS OF USEFUL LIFE
Building and structures,	
Masonry, brick, steel	30–50
Frame	20–35
Boilers, steam	17–20
Engines, steam and internal combustion	17–25
Electric generators and motors, large	20–25
Electric generators and motors, small	15
Locomotives, electric and internal combustion	10–12½
Mine cars, steel	10
Mine cars, wood	5
Mining machines	12½
Railroad ties	7–14
Railroad rails	20
Railroad frogs and switches	14
Automobiles and trucks	3– 6
Livestock	5–10
General machinery	10–20
Instruments and office machines	6–15

He notes that in valuation work special attention should be given to the condition of equipment and its adequacy to perform the service required. Account should be taken of the occurrence of obsolescence and of the fact that all equipment should be depreciated to salvage values within the remaining life of the property.

While depreciation charges for a mine plant should be based on the expected useful life of the various classes of structures and equipment involved, it is the opinion of the Committee that the aggregate of these charges should be distributed over the entire tonnage to be recovered by said plant and become a part of the per-ton cost of mining each ton of coal. This method is preferable to a fixed annual depreciation charge in that it avoids fluctuations in the depreciation item of mining cost due to variation in output.

Plant Salvage Value

Related closely to plant depreciation is the question of salvage value of plant structures and equipment and the relation it bears to original cost. Several expressions of opinion on this question agree that the salvage value of a mine plant at the end of its life rarely amounts to 5 per cent of its cost. Usually only a small salvage value resides even in such portable equipment as motors, cars and mining machines, owing to lack of standardization among mines, to variability of conditions at different operations and to continuous improvements in design of machinery. Structures and heavy equipment usually cost more to disassemble, move and re-erect than they are worth at their new location.

Mine Town

Dwellings, buildings of public assembly, water and lighting systems, improved streets and similar necessities or conveniences provided by a mining company for its employees should be classed ordinarily as part of its plant and appraised as such. Occasionally, however, a mining town is located in or near a general industrial or residential community that promises a useful life for the town independent of the mine it was built to serve and more enduring than that mine. Under these conditions cognizance should be taken of its independent earning power during the life of the mine, also of its sale value when the mine has exhausted the tributary coal, and its normal value as a component of the mine plant increased accordingly.

Surface

Surface land owned by a mining company frequently presents to the appraiser a problem similar to the one just mentioned for the company town. Where the company owns only the surface acreage necessary for the location of its plant, railroad sidings, town and other requirements of its mining activities, such surface becomes an essential part of its operating unit and is included in the general valuation on the basis of earnings. Under such conditions it makes but little difference what value is assigned to it.

Where surface suitable for plant sites is abundant, regional unit sale prices for similar land is a suitable gage. Where such surface is scarce an extra value may be assigned to represent the appraiser's judgment of the special worth of this surface for the particular purpose for which it is used.

However, it frequently happens that a coal property carries far more surface land than is needed for the prosecution of mining. In such cases the excess surface should be valued at what it may be expected to bring in the open market, giving due consideration to the possibilities of future damage from mining operations.

Mine Development

There is perhaps no feature of a developed mining property about which less unanimity exists among engineers and operators than Mine Development, what it includes and how it should be valued. Several members of the Committee have discussed this subject generally and Mr. Hicks has given it special consideration.

Mine Development, as here used and applied to a coal mine, comprises such portions of the underground workings as are essential to the production of the mine's normal output. It begins at the point of entry into the coal seam and includes entries or headings, air courses, break-throughs, permanent overcasts and drainage ways, room necks and the rooms themselves up to a point where a sufficiently large output is obtainable to bring the cost of mining approximately to normal.

In the case of a drift mine where development work starts simultaneously with plant construction, such a point is usually reached when the potential output is around 75 per cent of the designed daily tonnage.

With a shaft or slope mine, where the plant and railroad connections were being built as the shaft or slope was sinking, the net cost of mining after the seam is entered more rapidly approaches normal than in the case of a drift mine, because of better preparation and shipping facilities for the coal produced. Hence, for shaft or slope operations true development work may cease when the output is as low as 60 per cent of daily capacity.

Development does not include the means of access to the coal, such as shafts, shaft bottoms, slopes or drift mouths. It is, however, chargeable with the cost of rock removal from the entry roof or floor until the mine has been expanded to 75 per cent, say, of its required size.

Valuation of mine development is based on the same principle as plant or town valuation; namely, the cost of reproduction less depreciation. This cost of reproduction comprises all expenses of opening out the mine workings from the point of entry into the coal seam until they have reached the limits previously defined, less credits from the sale of coal recovered from these workings. It should be noted, however, that such expenses do not include the cost of mine track, machinery, etc., used in the development work unless they are of no further service in the subsequent operation of the mine.

When the cost or value of such development (which may be called the initial mine unit) has been established, it is then subject to amortization at a rate that will extinguish it by the time the mine has reached the boundaries of its coal area and production commences to decline. It is true that sufficient entries and working places must be maintained throughout this time to afford the normal mine output, so, theoretically, there should be no amortization of the development item until the workings cease to advance. Nevertheless, it is the opinion of this Committee that, in practice, it is proper to amortize this item over the mine life, as stated above, so that this charge will be relatively light as the mine nears exhaustion when operating and maintenance expenses are at their peak.

When the initial mine unit has been completed, this development asset of the property has its maximum value, which is thereafter gradually reduced by depreciation unless subsequently augmented by special charges. Two examples of such special charges may be mentioned:

1. An old mine may have certain excess or reserve development, such as entries driven into unworked or only partly worked territory, which is not needed at present to maintain the normal output. The measure of the value of such reserve development is the amount by which the normal cost of mining its tributary coal exceeds the estimated cost of mining that coal after giving effect to the savings occasioned by having the use of the excess development under consideration.

2. If after the mine has passed its initial development stage the workings encounter abnormal conditions, such as large faults or rock rolls, particularly weak roof, or excessive water flow, the extra expense of overcoming these obstacles may be charged to development and increase its value to the property.

Coal Reserves

Coal acreage to be considered in the appraisal of a developed property is the area of seam that will be mined within the life of the existing plant with its normal maintenance and replacements. There is, of course, no construction-cost basis of valuation for coal acreage as there is for plant, town and development; and usually there is no generally recognized purchase-and-sale value for it, as in the case of surface land. Therefore the worth of the coal in a developed property may best be measured by the value that remains after deducting from the total valuation of the property as a whole (determined from estimated earnings, as already indicated) the aggregate of the individual appraisals of plant, town, necessary surface and mine development. This residual value covers not only the coal itself but also such intangible assets of the enterprise as sales contracts, special markets, a favorable labor situation or especially good transportation facilities, the possession of any of which should be reflected in higher net earnings for the property.

While the value of any given coal acreage, determined as above outlined, can be reduced to a value per acre or per ton, emphasis should be given to the fact that such a unit value cannot be applied arbitrarily to another area of coal land, even though the coal in the two properties may be identical in thickness, quality, accessibility and other physical features. Certain elements of the valuation calculations, such as life of operation and cost of mining plant, have little or no connection with the physical features of the coal seam. Hence it might well be that two adjoining areas of a uniform coal bed, developed by separate mines, would have very different values per ton or per acre.

Mine Life

The proper length of life for a coal mine was discussed by several members of the Committee. Their general conclusion is that the normal life of a coal mine is that period of operation for which money invested in the property will bring the maximum return.

The first point to be considered in this connection is the market for the coal; that is, how much of the coal in question can be sold annually and at what price? This involves consideration of competitive fuels and transportation costs, and leads to a conclusion as to the annual tonnage that can be sold to afford the most satisfactory gross return per ton. With this yearly tonnage decided upon, the next step is to determine the investment in mine plant, transportation and surface required to afford the annual output desired, which, of course, must not exceed the marketable tonnage mentioned above. Following this determination, the prospective operator would estimate his actual operating costs (exclusive of interest, taxes and depreciation) and then ascertain the cost of a coal acreage sufficient to give the life of operation affording the largest annual return on the total capital investment.

Assuming actual operating costs per ton to be fixed by local conditions, overhead costs of interest, taxes, depreciation and depletion are the controlling factors in determining the most profitable length of life for the mine. With a short life and small coal reserves interest and tax charges per ton are relatively low while plant amortization is high. Conversely, long life and a large acreage increases the cost of interest and taxes but reduces depreciation charging. The most economical life is the one in which the aggregate per-ton charge for these overhead expenses is at a minimum.

Applying these considerations to actual conditions in the coal fields of the United States, it is the opinion of the Committee that only under very exceptional conditions can a mine life of 40 years be justified. Indeed, a life of 25 years rarely is warranted on the grounds of maximum annual returns to the owner. Calculations based on the experience of typical coal operations in this country indicate that many, if not most, of them would have shown the highest return on their invested capital had their life been limited to 15 years, or even less.

Generally speaking, a property with an expensive plant mining low-priced coal can justify a longer life than one where these conditions are reversed.

Excess Coal Reserves

A natural corollary to the above discussion of the proper life of a developed coal property is the question of how to value the coal reserves owned in excess of those required to give the mine its normal life. This subject is of major importance and misapprehensions regarding it have frequently led in the past to grave errors on the part of both mine owners and appraisers.

This Committee is in emphatic agreement that coal reserves owned in excess of those required by a mine for its normal life, and which are not to be disposed of or developed by a new operation within, say, 30 years, have little or no present value.

Perhaps the most striking evidence in support of this statement is given by the compound interest table. It shows that if the carrying charges of interest and taxes for coal land be as low as 8 per cent per annum, the cost of such land will double itself on the owner's books every 9 years. Thus if a company buys at $50 per acre coal land which it will not use for 27 years, it must anticipate that such acreage will be worth $400 per acre when its development begins, assuming the cost of capital and taxes to be only 8 per cent annually The failure of many large coal companies to earn a fair return on the capital invested in them has been due in no small degree to the burden of vast areas of idle coal land which they owned.

Where coal reserves in excess of those required to give normal lives to existing mines can and will be sold or developed by new or enlarged operations in the reasonably near future, they may have a value comparable to that of similar developed acreage.

Should they be held for sale they would be valued in accordance with the principles of appraising undeveloped coal lands to be considered subsequently in this report. However, in such instances the appraiser must take cognizance of the fact that the development of additional tonnage from the lands sold may, by competing in a limited market, lower the price to be received for the coal produced from existing mines and reduce their value correspondingly.

Should the intention of the owner be to hold the excess coal acreage until some future time when it will be developed by new mines, the logical method of appraisal is to calculate from estimated earnings what will be its probable value when development begins and then discount that value to its worth at the present time. Observing the rapid decrease in values resulting from discounting future capital sums at conservative carrying charges and considering the impossibility of forecasting with any assurance of accuracy the earnings of a coal operation even 10 years hence, it is evident that only low and uncertain present values attach to coal lands destined to be idle for as much as three decades.

Interest Rates

Interest rates are important factors in calculations of coal property values and appreciable differences of opinion exist among appraisers as to what rates are applicable under the varying conditions that affect the estimated earnings of a mine. The principal reason for varying the rate at which earnings are capitalized is to allow for the element of risk or hazard in all mining operations. Naturally, the greater the uncertainty attaching to conditions governing future earnings, the higher the return that should be allowed to capital subject to the risk.

There are two general methods of varying the capitalization of expected income in accordance with the risk involved. One is to capitalize the prospective earnings at ordinary commercial rates of interest and then reduce the principal sum thus obtained by an amount that represents in the judgment of the appraiser the hazard involved in the case at hand. The other is to select for use in the calculation that particular interest rate which in the opinion of the appraiser properly allows for the degree of uncertainty with which his estimates of earnings are attended. As it is true that the greater the element of risk the higher the rate of return that should be allowed to the capital invested, it follows that the capitalized value of an estimated income, or the appraised value of the property producing that income, decreases as the risk and assigned interest rate increase.

The method recommended by this Committee is the second of the two just mentioned; namely, varying the interest rate in capitalizing earnings to comport with the risk involved.

Chief among the hazards that usually must be assumed by capital invested in a developed coal property are:

1. Unexpected deterioration of the seam in minability or quality of its coal.
2. Damage to plant or mine from fire or explosion.
3. Interruption to mining by labor trouble and general industrial depression.
4. Local increase in mine wages.
5. Local increase in freight rates.
6. Increased costs due to legislation.
7. Declining market demand or prices for coal due to competition from other coal or from oil or gas.

In certain instances income from capital invested in a coal property is almost free from such risks as are enumerated above. Therefore it is subject to capitalization for valuation purposes at an interest rate approximating that applying to a sound real estate investment. An example would be the case where income is derived from minimum royalties of less than the regional rate paid by a lessee of established responsibility with a valuable plant subject to forfeiture in the event of default in royalty payments and where the life of the leasehold is of moderate duration. A proper capitalization rate in such an instance would ordinarily be 6 per cent per annum.

Many degrees of capital hazard will occur to the experienced coal property appraiser, varying from the low-risk instance just cited to the speculative enterprise where an operator is attempting to introduce a relatively poor coal into a highly competitive market.

No attempt has been made by this Committee to establish proper interest rates for the wide variety of conditions that confront appraisers of coal properties. It would, however, record its opinion that a fair rate at which to capitalize for valuation purposes prospective earnings of an owner-operator with a well equipped property having relatively good coal and mining conditions in one of the major coal fields of the United States is 12 per cent per annum.

In calculations of values from earnings another rate must be used besides the interest return on capital; namely, the sinking fund rate, or the interest allowed on the portion of earnings set aside periodically to redeem the capital investment at exhaustion of the property. It is almost universally agreed among appraisers that the proper rate for this purpose is 4 per cent per annum.

Coal Recovery

The percentage of extraction of coal by a mine is an important element in the appraisal of a coal property as it affects both the recoverable tonnage in the property and the life of operation. Mr. Duncan, of this Committee, has made an exhaustive search of the reports of various Federal and state investigators on this subject and has embodied his findings in an extensive and valuable set of tables. From them the following conclusions are presented:

Complete extraction of the coal in a mining area is never obtained in practice except in certain rare instances of longwall mining. Considering all the elements involved, the best extraction that may be expected with the room-and-pillar system of mining generally used in this country is 90 per cent, and that only in the case of a seam of high-grade coal with fine natural mining conditions and under a surface but little damaged by settling. Rarely indeed are such conditions found; so that in the principal bituminous coal fields of the Appalachian region, which afford most of the country's tonnage, recoveries of 70 to 80 per cent are the rule, with mines in many districts where the coal is of medium or poor grade recovering only about 60 per cent of their tributary tonnage.

In the Interior province and most of Ohio, where mine pillars are seldom drawn, recovery rarely exceeds 55 per cent and frequently falls below 50 per cent.

In the Anthracite region of Pennsylvania, recoveries range from 60 to 70 per cent in most mines, rising occasionally to 80 per cent where conditions are especially favorable.

UNDEVELOPED PROPERTIES

This class of properties presents so large a variety of conditions that no single method of appraisal can be selected as the best for the class as a whole. It is the opinion of the Committee that in situations where regional mining and marketing conditions have been established by developed coal properties, and where estimates of future earnings for the undeveloped lands in question can be made with reasonable assurance, the method of valuation by capitalizing prospective earnings is the best.

Future earnings may accrue to the owner either from direct operation of the property or through royalties from a lessee. Therefore the appraiser is frequently aided in establishing his basic factors for the valuation calculations by a knowledge of current royalties paid on similar lands in the region.

In applying the prospective earnings method of valuation the appraiser must remember to discount back to the present whatever value is obtained by capitalizing earnings, for that value applies only at the time when the income begins. Furthermore, it must be remembered that no income that will not be received in less than 40 years is suitable for use in valuation calculations.

In a new coal territory where commercial mines have not been established and where little is known of future labor conditions, transportation service, freight rates or markets the aforementioned method of appraisal is not applicable. Under such conditions the only value that can be assigned to a particular coal area is the one based on the speculative prices established by current sales of similar areas in the region. At best this is but a poor method. In the first place it is usually impossible to find a recently sold tract similar in many essential respects to the one under consideration and, secondly, such prices often fluctuate widely over a short period of years and are affected by industrial and financial conditions entirely unrelated to the inherent value of the coal land itself.

Consideration must be given to the size and location of the property so appraised, for it is obvious that a unit price of $50 an acre might have been paid for a 400-acre coal tract fronting on a stream along which a railroad was expected to come, whereas this would be a wholly unwarranted per-acre value for the whole of a 4000-acre boundary, even though it adjoined the smaller tract and carried the same coal seam.

Scarcity of exposures of the principal coal bed in a property may occasion doubt as to its persistency in section and quality and uncertainty as to the quantity of commercial coal available. This may be true of either an undeveloped or a developed property, though it is more apt to be encountered in an undeveloped area. Wherever it is felt it should be allowed for in the appraisal.

Where the seam is known regionally to be uniform in character and when the few exposures on or near the property to be appraised show it of ample mining thickness, then the allowance for uncertainty of commercial tonnage need not be large. But where these conditions do not obtain, thorough prospecting should be required before more than a nominal value is assigned to the property.

Relative Value of Mine and Management

By Eli T. Conner

Almost irrespective of quantity or quality of mineral contained in a property, its economic value as a mineral reserve may be set down as 30 per cent of its whole value and the rest, or 70 per cent, can be regarded as determined by the ability of the operator as a manager and his established qualifications for marketing his product. Any business enterprise is dependent for 75 per cent of its success on management and 25 per cent on the property.

Values should be based on the "measuring stick" longest applied to coal properties, a fair royalty per ton. Valuation of developments, improvements, plant and equipment of a going enterprise should be largely discounted below reproduction value at date of appraisal, because capital thus invested must ultimately be returned out of profits from mineral extracted, processed and marketed. Plant and equipment, though necessary for prosecution of an enterprise, constitute a liability rather than an asset.

Depreciation Rates and Recovery Percentages

By W. G. Duncan, Jr.

Depreciation rates cannot be established at any fixed or arbitrary values for all coal properties. Extraordinary obsolescence, due to radical changes in mining methods and coal preparation and to invention, must also be considered. The figures of Table 1 were compiled from various sources and may be of some value as a rough check on depreciation rates. These were prepared for accountants rather than for engineers. Items marked "1" are taken from "Depreciation Studies, Preliminary Report of the Bureau of Internal Revenue." Items marked "2" are derived from "The Principles of Mine Valuation," by A. W. Hesse, and represent the average of the range of values in that table. Items marked "3" come from various sources.

Table 1.—*Probable Useful Life and Rate of Depreciation of Coal Mine Equipment*

Item	Probable Useful Life, Years	Depreciation Rate, Per Cent	Authority
Adding machines	10	10	1
Ammeters, recording	15	6⅔	1
Ash-handling equipment	13	7½	1
Automobiles	4	25	1
Awnings	5	20	1
Ballast RR	25	4	1
Barns: frame	25	4	1
masonry	50	2	1
Batteries: heavy type	15	6⅔	1
light type	10	10	1
Bins: concrete and brick	20	5	2
steel	20	5	2
wood	10	10	2
Belting	6⅔	15	3
Blacksmith shop equipment	15	6⅔	1
Boats: motor	5	20	2
row	10	10	2
Blowers	15	6⅔	1
Blue-printing machines	15	6⅔	1
Boilers:			
Over 50 hp., fire tube horizontal	20	5	1
Under 50 hp., fire tube horizontal	17	6	1
Over 50 hp., fire tube locomotive	17	6	1
Under 50 hp., fire tube locomotive	14	7	1
Under 50 hp., fire tube portable	10	10	1
Over 50 hp., water tube under 225 lb. pressure	25	4	1
Over 50 hp., water tube over 225 lb. pressure	22	4½	1
Boiler fittings	18	5½	1
Bonding, track	12½	8	2
Bookkeeping machines	6	16⅔	1
Breakers: concrete or steel	33	3	1
wood	15	6⅔	1
Breeching and flues	22	4½	1
Briquetting machinery	15	6⅔	1
Bridges: masonry or concrete	60	1⅔	1
steel	40	2½	1
timber	16	6¼	1
Buckets, clamshell	12	8⅓	2

Estimates of extraction percentages should be based on results of past operation. But if no accurate information is available, the Tables 2, 3 and 4 may be taken as a guide. They were taken from reports of the United States Coal Commission and bulletins for the States of Pennsylvania, Ohio, Illinois, Alabama and Tennessee, published thereafter. The report by counties of the percentage extraction in Pennsylvania is from Pennsylvania Topographic and Geologic Survey *Bulletin* M–6, Part 3, 1928.

TABLE 1.—(*Continued*)

Item	Masonry, Brick, Concrete, Reinforced Concrete, Brick and Steel, Steel Frame, Steel and Stucco (Fireproof)		Masonry, Slow-burning, with or without Steel Frame		Masonry, with Frame Interior		Frame		Authority
	Probable Useful Life, Years	Depreciation Rate, Per Cent	Probable Useful Life, Years	Depreciation Rate, Per Cent	Probable Useful Life, Years	Depreciation Rate, Per Cent	Probable Useful Life, Years	Depreciation Rate, Per Cent	
Buildings:									1
Dwellings:									
1 family	50	2	50	2	50	2	33	3	
2, 3, or 4 family	45	2¼	40	2½	33	3	30	3⅓	
Factories	40	2½	35	$2\frac{6}{7}$	30	3⅓	25	4	
Foundries	50	2	40	2½	28	3½	25	4	
Garages:									
private	50	2	40	2½	40	2½	25	4	
public	50	2	35	$2\frac{6}{7}$	30	3⅓	20	5	
Hotels and elevator apartments	35	$2\frac{6}{7}$	30	3⅓	25	4	22	4½	
Houses. See Houses—miners									
Loft buildings	45	2¼	35	$2\frac{6}{7}$	30	3⅓	25	4	
Mill-type buildings	40	2½	35	$2\frac{6}{7}$	30	3⅓	20	5	
Machine shops	40	2½	33	3	28	3½	25	4	
Office buildings	40	2½	35	$2\frac{6}{7}$	30	3⅓	25	4	
Power stations	50	2	40	2½	33	3	25	4	
Roundhouses	50	2	40	2½	33	3	28	3½	
Row houses	45	2¼	40	2½	35	$2\frac{6}{7}$	30	3⅓	
Stores	50	2	40	2½	35	$2\frac{6}{7}$	28	3½	
Stores, 1 or 2 stories, or rooms or apartments	40	2½	35	$2\frac{6}{7}$	30	3⅓	25	4	
Theaters	33	3	25	4	22	4½	20	5	
Warehouses	50	2	50	2	45	2¼	35	$2\frac{6}{7}$	
Warehouses, skeleton pier and special commodity warehouses, cold storage and packing	40	2½	33	3	28	3½	20	5	

TABLE 1.—(*Continued*)

Item	Probable Useful Life, Years	Depreciation Rate, Per Cent	Authority
Cabinets and files	15	6⅔	1
Cages: car	15	7	1
self-dumping	5	20	1
Carpenter-shop equipment	20	5	1
Carriers: feed	20	5	1
hay	25	4	1
litter	5	20	1
Cars, mine: steel	10	10	1
wood	5	20	1
Cases: book	20	5	1
display	20	5	1
Cattle, breeding or dairy	8	12½	1
Chairs: bentwood	5	20	1
heavy	16	6¼	1
Check writers	8	12½	1
Chimneys: concrete or brick	50	2	1
steel, self-supporting lined	28	3½	1
steel, self-supporting unlined	22	4½	1
Cisterns	33	3	1
Classifiers	10	10	1
Clocks: time	15	6⅔	1
wall	20	5	1
time-stamping	10	10	1
Coal-cutting machines	12½	8	1
Compensators, starting	15	6⅔	1
Compressors: portable	13	7½	1
stationary	20	5	1
Comptometers	10	10	1
Condensers: steam atmospheric	25	4	1
steam surface	20	5	1
Conduits: concrete and vitrified	50	2	1
fiber or steel	25	4	1
Conveyors	12½	8	1
Coolers, water	10	10	1
Cranes: jib	15	6⅔	1
locomotive and tractor	18	5½	1
traveling, gantry and bridge	20	5	1
Crushers: gyratory or jaw	15	6⅔	1
roll	10	10	1
Culverts, railroad: concrete	60	1⅔	1
steel	20	5	1
wood	14	7	1
Cutting and welding apparatus, oxyacetylene	10	10	1
Dams: earthen, concrete or masonry	75	1⅓	1
Derricks: steel	13	7½	2
wood	6⅓	15	2
Desks	15	6⅔	1
Dictaphones	6	16⅔	1
Drag-line excavators (light)	10	10	1
Drills, rock, air or steam, mounted	5	20	1
Drill presses	10	10	2
Driers, sand	10	10	3
Doors, mine, patent	5	20	2
Dumps: crossover	10	10	1
rotary: heavy	15	6⅔	1
light	10	10	1
Economizers	15	6⅔	1
Electric-lighting systems	20	5	3
Elevators: bucket	10	10	1
freight	25	4	1
Engines: haulage	12½	8	1
diesel	20	5	1
gas and gasoline	17	6	1
steam high-speed	16	6¼	1
steam low-speed	25	4	1
Exciters	20	5	1
Fans: electric	10	10	1
ventilating	13	7½	2
Fences: masonry	25	4	1
steel posts and wire	15	6⅔	1
wood posts and wire	12	8⅓	1
wood	14	7	1
Filters, oil	20	5	1
Fire equipment:			
Fire-alarm systems	25	4	1
Movable equipment	15	6⅔	1
Fuel systems pulverized coal	20	5	1
Furniture, boarding house	10	10	1
Framers, timber	15	6⅔	1
Frogs and switches (railroad)	14	7	1
Gages, recording	15	6⅔	
Generators, alternators and motors:			1
Above 3000 kva	28	3½	1
1000 to 3000 kva	25	4	1
Units from 50 hp. to 1000 kva.			
high-speed, direct-connected	20	5	1
high-speed, belt-driven	17	6	1
low-speed, direct-connected	25	4	1
low-speed, belt-driven	22	4½	1
Units below 50 hp.			
alternating current	17	6	1
direct current	14	7	1
Harness	10	10	3
Headframes: steel	30	3⅓	1
wood	15	6⅔	1
Heaters. feed water: closed type	25	4	1
open type	22	4½	1
Heating systems:			
Boilers and furnaces	20	5	1
Radiators	25	4	1
Hoists: air tugger	7	14	1
gasoline	10	10	1
portable electric	10	10	1
stationary electric or steam	20	5	1
Horses, breeding or work	10	10	1
Hospital equipment	15	6⅔	1
Houses, miners:			
box type	15	6⅔	1
ceiled and weatherboarded	25	4	1
lathed and plastered	30	3⅓	1
masonry	40	2½	1
Jigs	12½	8	1
Larries	15	6⅔	1
Lighting systems:			
Conduit, fittings and wiring	25	4	1
Fixtures	15	6⅔	1
Lightning arresters, station type	20	5	1
Loaders: box-car	10	10	1
mechanical	7	14	1
Lockers	20	5	1
Locomotives:			
electric:			
storage battery	10	10	1
trolley	12½	8	1
gasoline and oil	10	10	1
steam:			
gear drive	15	6⅔	1
rod drive	25	4	1
Machines: calculating	5	20	2
keyseating	13	7½	2
mining	12½	8	2
pipe-threading	15	6⅔	2
refrigerating	20	5	2
shearing	18	5½	2
welding	10	10	2
Machine-shop equipment	20	5	1
Meters: electric	20	5	2
recording steam or water	15	6⅔	1

TABLE 1.—(*Continued*)

Item	Probable Useful Life, Years	Depreciation Rate, Per Cent	Authority
Mining machines	12½	8	1
Mimeograph machines	10	10	1
Mixers, concrete	5	20	2
Motor, see generators			
Mules: work	10	10	1
mine	5	20	1
Pavements: asphalt	12½	8	1
brick	14	7	1
concrete	20	5	1
macadam	15	6⅔	1
plank	6	16⅔	1
Plows	12	8⅓	1
Plumbing:			
fixtures	25	4	1
pipes, brass or copper, same as building			
iron, cold water	30	3⅓	1
iron, hot water or steam	25	4	1
sewer, cast iron or vitrified same as building			
Poles: concrete	50	2	1
iron and steel	33	3	1
wood, creosoted	22	4½	1
wood, dipped butt	20	5	1
wood, in earth	15	6⅔	1
Presses: hay-baling	12	8⅓	1
drill	10	10	2
hydraulic	17	6	2
Pumps, power plant:			
auxiliary	18	5½	1
centrifugal and rotary	20	5	1
direct acting	25	4	1
gear-driven	22	4½	1
Railroad: grading	33	3	3
track-laying and surfacing	20	5	3
Rails, with fastenings and joints	20	5	1
Refrigerating machines	20	5	1
Rescue helmets	6	16⅔	1
Ropes: cage	2	50	2
incline	3	33⅓	2
Safes and vaults	40	2½	1
Scales: counter	10	10	1
platform, dormant	20	5	1
railroad	20	5	1
Scrapers: slip	2	50	1
wheel	10	10	1
Screens	10	10	1
Separators, spiral, coal	10	10	1
Separators, steam	25	4	1
Sheds, frame	25	4	1
Shovels, electric or steam	15	6⅔	1
Silos: concrete	40	2½	1
metal	25	4	1
wooden	18	5½	1
Skips, hoisting	5	20	1
Soda fountains:			
Carbonators	10	10	1
Chairs	10	10	1
Mirrors	20	5	1
Refrigeration, new type	20	5	1
Serving bars	12	8⅓	1
Silverware	5	20	1
Tables	10	10	1
Stable equipment	10	10	3
Stand pipes: concrete	50	2	1
steel	30	3⅓	1
Stokers	12½	8	2
Switchboards and wiring	20	5	1
Tables, concentrating	12½	8	1
Tanks: galvanized iron	10	10	1
steel	25	4	1
wood	15	6⅔	1
Telephone, instruments	12½	8	1
Telephone lines	10	10	3
Telephones, mine	5	20	2
Thickeners: concrete	33	3	1
steel	15	6⅔	1
Ties, railroad:			
Cedar	12½	8	1
Fir, pine and tamarack	8	12½	1
Hemlock and mixed oak	7	14	1
Steel	11	9	1
Treated	14	7	1
White oak	11	9	1
Timbering: steel	10	10	2
Tipples: steel	30	3⅓	1
wood	15	6⅔	1
Towers, cooling (frame)	15	6⅔	1
Tractors	6	16⅔	1
Tramways, aerial	15	6⅔	1
Transformers	20	5	1
Traps, steam	20	5	1
Trestles, timber	14	7	1
Trucks:			
Electric	10	10	1
Gas: under 1000	3	33⅓	1
1000 to 1500	4	25	1
1500 to 2500	5	20	1
over 2500	6	16⅔	1
Turbines: hydraulic	40	2½	1
steam	22	4½	1
Typewriters	6	16⅔	1
Wagons	7	14	1
Washers, coal-mining	13	7½	2
Wires: feeder	25	4	2
trolley	25	4	2
Wiring:			
Aluminum or bare copper	40	2½	1
Copper insulated	16	6¼	1
Iron	15	6⅔	1
Trolley	20	5	1

TABLE 2.—*Extraction from Coal Fields East of the Mississippi*

Bed	Locality	Total Loss, Per Cent	Actual Extraction, Per Cent	Avoidable Loss, Per Cent	Unavoidable Loss, Per Cent	Extraction That Should be Obtained, Per Cent	Remarks
			ALABAMA[a]				
Black Creek Warrior coal field.	Blount, Cullman, Etowah, Jefferson, Marion, Walker and Winston counties	38.75	61.25	22.50	16.25	83.75	
Jagger seam Warrior field.	Carbon Hill, Townley and Walker counties	44.70	55.30	20.5	24.2	75.8	
Mary Lee Warrior field.	Jefferson and Walker counties	39.15	60.85	23.50	15.65	84.35	
Pratt seam Warrior field.	Jefferson and Walker counties	12.40	87.60	3	9.40	90.60	
Thompson seam Cahaba coal field	Towns of Acton, Roebuck, Colemont, Garnsey, Coleanor, Piper and Blocton	51.30	48.70	35.5	15.8	84.2	
			ILLINOIS[b]				
No. 2 bed, "Third Vein," Northern Ill.	Lasalle, Bureau, Marshall and Grundy counties	5	95	5	0	100	Longwall system used.
No. 5 bed.........	Peoria, Fulton and Knox counties	53.0	47.0	28	25.0	75.0	Pillars not drawn under streams and railroads.
No. 6 bed.........	Central and Belleville districts	47.5	52.5	26	21.5	78.5	Pillars not drawn because of valuable surface land.
No. 5 and No. 6...	Springfield district. Logan, Macon, Menard and Sangamon counties	51	49	26	25	75.0	Pillars not drawn because of valuable surface land.
No. 6 and No. 2....	Southern Ill. Includes Williamson, Franklin, Jackson, Perry and Washington counties	53	47	36.0	17.0	83.0	Pillars not entirely drawn.
No. 5............	Saline county	46	54	31	15	85.0	Land valuable. Pillars not drawn.
No. 6 and No. 7...	Danville district	43	57	23	20	80	Many pillars not drawn.

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[a] Authority: J. J. Forbes: Coal Losses in Alabama. *Bulletin* No. 1, Alabama State Mine Experimental Station. Also published in Part III, page 1854, Report of U. S. Coal Commission, 1925.

[b] This information was obtained from Cooperative Mining Series *Bulletin* No. 30, "Coal Lossse in Illinois," by C. A. Allen. Also published in Part III of Report of U. S. Coal Commission.

TABLE 2.—(*Continued*)

Bed	Locality	Total Loss, Per Cent	Actual Extraction, Per Cent	Avoidable Loss, Per Cent	Unavoidable Loss, Per Cent	Extraction That Should be Obtained, Per Cent	Remarks
			INDIANA[c]				
	Clinton-Brazil	44	56.0	26	18	82.0	It seems from the large percentage of coal lost in pillars that they are not drawn.
	Sullivan-Linton........	43.0	57.0	30.0	13	87.0	
	Knox	37	63.0	27.0	10	90.0	
	Southern	40	60.0	25.0	15.0	85.0	
			KENTUCKY[c]				
	Eastern:						
	Northeast	23	77	2.7	20.3	79.7	
	Hazard	30	70	6	24	76.0	
	Harlan	27.8	72.2	10.8	17	83.0	
	Western	37.0	63.0	20	17	83.0	Pillars not drawn.
			MARYLAND[c]—Pillars drawn				
	Georges Creek	31.4	68.6	11.9	19.5	80.5	
	Upper Potomac	26.7	73.3	15.3	11.4	88.6	
			OHIO[d]				
U. Freeport No. 7..	Lisbon-Palmyra dist. Columbiana, Carroll and Portage counties	43.0	57.0	26.0	17.0	83.0	No pillars drawn.
M. Kittanning No. 6		51.0	49.0	35.0	16.0	84.0	No pillars drawn.
L. Kittanning No. 5		43.0	57.0	29.0	14.0	86.0	No pillars drawn.
Sharon No. 1.......		21.0	79.0	10.0	11.0	89.0	Pillars drawn.
M. Kittanning No. 6	Massilon district, Stark, Wayne, Summit and Portage counties	44.0	56.0	12.0	32.0	68.0	No pillars drawn.
L. Kittanning No. 5.		46.0	54.0	33.0	13.0	87.0	No pillars drawn.
Brookville No. 4...		30.0	70.0	17.0	13.0	87.0	No pillars drawn.
Mercer No. 3......		53.0	47.0	38.0	15.0	85.0	No pillars drawn
Sharon No. 1......		24.0	76.0	11.0	13.0	87.0	Pillars drawn.
U. Freeport No. 7..	Coshocton-Goshen dist. Carroll, Tuscarawas, Coshocton and Holmes counties	38.0	62.0	25.0	13.0	87.0	No pillars drawn.
M. Kittanning No. 6		47.0	53.0	31.0	16.0	84.0	No pillars drawn.
L. Kittanning No. 5.		47.0	53.0	30.0	17.0	83.0	No pillars drawn.
Mercer No. 3......		54.0	46.0	37.0	17.0	83.0	No pillars drawn.
Sharon No. 1......		29.0	61.0	15.0	14.0	86.0	No pillars drawn.
Pomeroy..........	Belmont dist.						
Meigs Creek.......							Not much pillar drawn.
Pittsburgh........							
		40.0	60.0	25.0	15.0	85.0	

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[c] Data found in Part III, Table 5, Report of U. S. Coal Commission, 1925.

[d] Authority: James D. Sisler and C. A. Allen: Coal Losses in Ohio. Ohio State Univ. *Bulletin* No. 29 (1929).

TABLE 2.—(*Continued*)

Bed	Locality	Total Loss, Per Cent	Actual Extraction, Per Cent	Avoidable Loss, Per Cent	Unavoidable Loss, Per Cent	Extraction That Should be Obtained, Per Cent	Remarks
			OHIO[d]—(*Continued*)				
Pittsburgh No. 8	Cambridge dist. Guernsey and Noble counties	46.0	54.0	12.0	34.0	66.0	No pillars drawn.
U. Freeport No. 7		39.0	61.0	16.0	23.0	77.0	No pillars drawn.
M. Kittanning No. 6		37.0	63.0	25.0	12.0	88.0	No pillars drawn.
L. Kittanning No. 5		44.0	56.0	33.0	11.0	89.0	No pillars drawn.
U. Freeport No. 7	Crooksville (Muskingum) district. Muskingum and Perry counties	46.0	54.0	29.0	17.0	83.0	
M. Kittanning No. 6		28.0	72.0	12.0	16.0	84.0	Pillars not drawn.
L. Kittanning No. 5		44.0	56.0	28.0	16.0	84.0	Pillars not drawn.
Clarion No. 4A		47.0	53.0	30.0	17.0	83.0	Pillars not drawn.
Pittsburgh No. 8	Jackson and Hocking district, Athens, Vinton, Perry and Jackson counties	47.0	53.0	13.0	34.0	66.0	Pillars not drawn.
U. Freeport No. 7		46.0	54.0	13.0	33.0	67.0	Pillars not drawn.
M. Kittanning No. 6		33.0	67.0	18.0	15.0	85.0	
L. Kittanning No. 5		45.0	55.0	31.0	14.0	86.0	
Clarion No. 4A		46.0	54.0	31.0	15.0	85.0	
Brookville No. 4		45.0	55.0	31.0	14.0	86.0	
Mercer No. 3		41.0	59.0	28.0	13.0	87.0	Pillars not drawn.
Quakertown No. 2		32.0	68.0	20.0	12.0	88.0	
Sharon No. 1		40.0	60.0	26.0	14.0	86.0	Pillars not drawn.
Pomeroy No. 8A	Ironton-Pomeroy district, Meigs, Gallia and Lawrence counties	43.0	57.0	32.0	11.0	89.0	No pillars drawn.
Pittsburgh No. 8		43.0	57.0	30.0	13.0	87.0	No pillars drawn.
M. Kittanning No. 6		38.0	62.0	25.0	13.0	87.0	Pillars lost by squeezes.
L. Kittanning No. 5		39.0	61.0	26.0	13.0	27.0	Pillars lost by squeezes.
Clarion No. 4A		43.0	57.0	31.0	12.0	88.0	Pillars lost by squeezes.
			PENNSYLVANIA[e]				
U. Freeport	Penn and Plum townships, Allegheny county	42.0	58.0	26.0	16.0	84.0	
Pittsburgh	Pittsburgh steam coal dist., Allegheny and Wash. counties	28.0	72.0	13.0	15.0	85.0	Limited pillar recovery.
Pittsburgh	Pittsburgh gas coal dist., Greene, Washington, Fayette, Allegheny and Westmoreland counties	22.0	78.0	10.0	12.0	88.0	
Mercer	Pittsburgh coking coal dist., southwest Pennsylvania, west of Chestnut Ridge						60 per cent recovery from beds below the Pittsburgh.
Brookville-Clarion							
L. Kittanning							
M. Kittanning							
U. Kittanning							
L. Freeport							
U. Freeport							
Pittsburgh		16.0	84.0	7.0	9.0	91.0	
Redstone			60.0				
Sewickley		22.0	78.0	10.0	12.0	88.0	
Waynesburg		31.0	69.0	16.0	15.0	85.0	

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[d] Authority: James D. Sisler and C. A. Allen: Coal Losses in Ohio. Ohio State Univ. *Bulletin* No. 26 (1929).

[e] Authority: J. D. Sisler: Bituminous Losses and Mining Methods in Pennsylvania. Topographic and Geologic Survey *Bulletin* M-4 (1924).

TABLE 2.—*(Continued)*

Bed	Locality	Total Loss, Per Cent	Actual Extraction, Per Cent	Avoidable Loss, Per Cent	Unavoidable Loss, Per Cent	Extraction That Should be Obtained, Per Cent	Remarks
PENNSYLVANIA[e]—*(Continued)*							
Brookville	Beaver-Mercer dist. Beaver, Lawrence, Mercer and Butler counties	37.0	63.0	20.0	17.0	83.0	
L. Kittanning		38.0	62.0	19.0	19.0	81.0	Pillars not drawn.
M. Kittanning		38.0	62.0	20.0	18.0	82.0	
U. Kittanning		40.0	60.0	22.0	18.0	82.0	
U. Freeport		31.0	69.0	20.0	11.0	89.0	
Brookville	Allegheny-Clarion dist.	36.0	64.0	25.0	11.0	89.0	
Clarion		49.0	51.0	34.0	15.0	85.0	
L. Kittanning		42.0	58.0	30.0	12.0	88.0	
U. Kittanning		62.0	38.0	51.0	11.0	89.0	No pillars drawn.
L. Freeport		42.0	58.0	21.0	21.0	79.0	Limited pillar recovery.
U. Freeport		44.0	56.0	29.0	15.0	85.0	
L. Kittanning	Ohiopyle-Ligonier dist., E. Fayette and Westmoreland counties	42.0	58.0	21.0	21.0	79.0	
U. Kittanning		32.0	68.0	16.0	16.0	84.0	
L. Freeport		55.0	45.0	32.0	23.0	77.0	
U. Freeport		24.0	76.0	11.0	13.0	87.0	
Pittsburgh		21.0	79.0	10.0	11.0	89.0	
Clarion	Indiana district, Indiana and Westmoreland counties		65.0				
L. Kittanning		34.0	66.0	18.0	16.0	84.0	
U. Kittanning			70.0				
L. Freeport		30.0	70.0	15.0	15.0	85.0	
U. Freeport		22.0	78.0	9.0	13.0	87.0	
Pittsburgh		29.0	71.0	16.0	13.0	87.0	
Brookville	Jefferson-Elk dist., Jefferson, Elk and Clearfield counties	26.0	74.0	12.0	14.0	86.0	
L. Kittanning		23.0	77.0	11.0	12.0	88.0	
U. Kittanning		20.0	80.0	7.0	13.0	87.0	
L. Freeport		23.0	77.0	9.0	14.0	86.0	
U. Freeport		21.0	79.0	9.0	12.0	88.0	
Brookville	Cambria district, Cambria and parts of Blair and Somerset counties	36.0	64.0	20.0	16.0	84.0	
L. Kittanning		25.0	75.0	12.0	13.0	87.0	
U. Kittanning		22.0	78.0	10.0	12.0	88.0	
L. Freeport		23.0	77.0	10.0	13.0	87.0	
U. Freeport		21.0	79.0	7.0	14.0	86.0	
Brookville	Somerset district, Somerset and part of Bedford county	30.0	70.0	17.0	13.0	87.0	
Clarion		30.0	70.0	17.0	13.0	87.0	No pillars drawn.
L. Kittanning		26.0	74.0	14.0	12.0	88.0	
U. Kittanning		26.0	74.0	13.0	13.0	87.0	
L. Freeport		29.0	71.0	13.0	16.0	84.0	
U. Freeport		26.0	74.0	10.0	16.0	84.0	
Pittsburgh		26.0	74.0	14.0	12.0	88.0	
Redstone		40.0	60.0	17.0	23.0	77.0	
Brookville	Clearfield dist., Centre, Clearfield and part of Cambria counties	36.0	64.0	23.0	13.0	87.0	
L. Kittanning		38.0	62.0	22.0	16.0	84.0	
M. Kittanning		33.0	67.0	14.0	19.0	81.0	
U. Kittanning		35.0	65.0	20.0	15.0	85.0	Limited pillar recovery.
L. Freeport		26.0	74.0	15.0	11.0	89.0	
U. Freeport		26.0	74.0	13.0	13.0	87.0	

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[e] Authority: J. D. Sisler: Bituminous Losses and Mining Methods in Pennsylvania. Topographic and Geologic Survey *Bulletin* M-4 (1924).

TABLE 2.—(*Continued*)

Bed	Locality	Total Loss, Per Cent	Actual Extraction, Per Cent	Avoidable Loss, Per Cent	Unavoidable Loss, Per Cent	Extraction That Should be Obtained, Per Cent	Remarks
PENNSYLVANIA[e]—(*Continued*)							
Fulton (Clar.).....	Huntingdon-Broadtop dist., Bedford, Fulton and Huntingdon counties	32.0	68.0	17.0	15.0	85.0	
Bartnett (L.K.)....		34.0	66.0	12.0	22.0	78.0	Pillars crushed.
Kelly (U.F.).......		30.0	70.0	11.0	19.0	81.0	Pillars crushed.
Bloss (L.K.?)......	Blossburg (Tioga) dist.	23.0	77.0	8.0	15.0	85.0	
Morgan (L.F.?)....		23.0	77.0	10.0	13.0	87.0	
Seymour (U.F.)....		37.0	63.0	13.0	24.0	76.0	
TENNESSEE[f]							
Clear Fork Mingo..	Claiborne County, Tenn., Bell County, Ky.	16.5	83.5	8.5	8.0	92.0	Pillars drawn possible.
Red Ash..........	Caryville dist., Campbell County	44.5	55.5	31.0	13.5	86.5	
Jellico............	Jellico dist., Campbell and Clayborne counties, E. Ky.	26.0	74.0	8.5	17.5	82.5	
Coal Creek........	Coal Creek dist., Anderson, Campbell and Morgan counties	36.5	63.5	20.0	16.5	83.5	
Bon Air No. 2.....	Crawford-Wilder dist., Fentress and Overton counties	18.5	81.5	11.5	7.0	93.0	
VIRGINIA[g]							
	S. Western Va.	20.0	80.0	5.0	15.0	85.0	
WEST VIRGINIA[h]							
	Panhandle	40.0	60.0	25.0	15.0	85.0	
	Fairmont	20.0	80.0	8.0	12.0	88.0	
	Preston-Taylor	23.0	77.0	11.0	12.0	88.0	
	Barbour-Upshur-Randolph	20.0	80.0	9.0	11.0	89.0	
	U. Potomac	26.7	73.3	15.3	11.4	88.6	
	Coal and coke	25.0	75.0	13.0	12.0	88.0	
	Mason-Putnam	35.0	65.0	19.0	16.0	84.0	
	Kanawha	33.0	67.0	17.0	16.0	84.0	
	Gauley	34.0	66.0	21.0	13.0	87.0	
	New River	32.0	68.0	17.0	15.0	85.0	
	Boone-Logan	19.0	81.0	7.0	12.0	88.0	
	Kenova-Thacker	30.0	70.0	18.0	12.0	88.0	
	Pocahontas	9.0	91.0	1.0	8.0	92.0	

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[e] Authority: J. D. Sisler: Bituminous Losses and Mining Methods in Pennsylvania. Topographic and Geologic Survey *Bulletin* M-4 (1924).

[f] Authority: J. J. Forbes: Coal Losses in Tennessee. *Bulletin* No. 33-E (1925).

[g] Authority: U. S. Coal Commission Report for 1925. Part III, Table 5.

[h] Authority: U. S. Coal Commission Report for 1925. Part III, Table 5.

TABLE 3.—*Percentage of Bituminous-coal Extraction in Pennsylvania*[a]

Bed	Locality	Extraction, Per Cent	Bed	Locality	Extraction, Per Cent
Pittsburgh	Allegheny County	93.9	Upper Freeport	Elk County	84.70
Thick Freeport	Allegheny County	78.5	Lower Freeport	Elk County	89.70
Thin Freeport	Allegheny County	59.68	Lower Kittanning	Elk County	86.20
Redstone	Allegheny County	53.17	Clermont (Clarion)	Elk County	76.10
Pittsburgh	Armstrong County	95.5	Alton	Elk County	50.00
Upper Freeport	Armstrong County	77.89	Waynesburg	Fayette County	64.00
Lower Freeport	Armstrong County	64.62	Sewickley	Fayette County	68.83
Upper Kittanning	Armstrong County	56.62	Redstone	Fayette County	50.00
Lower Kittanning	Armstrong County	66.55	Pittsburgh	Fayette County	86.00
Brookville	Armstrong County	52.82	Freeport	Fayette County	49.28
Pittsburgh	Beaver County	78.8	Upper Kittanning	Fayette County	68.36
Upper Freeport	Beaver County	54.43	Lower Kittanning	Fayette County	67.00
Lower Freeport	Beaver County	50.00	Waynesburg	Greene County	64.52
Middle Kittanning	Beaver County	51.73	Sewickley	Greene County	73.29
Lower Kittanning	Beaver County	51.50	Pittsburgh	Greene County	73.21
Brookville	Beaver County	50.00	Freeport	Greene County	42.50
Upper Freeport	Blair County	67.20	Pittsburgh	Indiana County	92.70
Lower Freeport	Blair County	50.00	Upper Freeport	Indiana County	75.12
Upper Kittanning	Blair County	50.00	Lower Freeport	Indiana County	69.34
Lower Kittanning	Blair County	90.10	Lower Kittanning	Indiana County	71.32
Brookville	Blair County	73.50	Upper Freeport	Jefferson County	75.60
Upper Freeport	Broad Top coal field, Bedford Fulton and Huntingdon counties	77.00	Lower Freeport	Jefferson County	84.20
Lower Kittanning		70.10	Upper Kittanning	Jefferson County	63.20
Clarion		69.80	Middle Kittanning	Jefferson County	67.10
Lower Kittanning	Bradford County	88.80	Lower Kittanning	Jefferson County	65.80
Brookville	Bradford County	51.00	Brookville	Jefferson County	72.36
Upper Freeport	Butler County	59.27	Upper Freeport	Lawrence County	54.00
Lower Freeport	Butler County	46.93	Lower Freeport	Lawrence County	51.35
Upper Kittanning	Butler County	55.79	Middle Kittanning	Lawrence County	51.05
Middle Kittanning	Butler County	49.04	Lower Kittanning	Lawrence County	50.61
Lower Kittanning	Butler County	49.79	Brookville	Lawrence County	51.04
Clarion	Butler County	48.87	Mercer	Lawrence County	40.00
Brookville	Butler County	60.52	Upper Freeport	Lycoming County	62.50
Upper Freeport	Cambria County	77.30	Middle Kittanning	Lycoming County	75.00
Lower Freeport	Cambria County	73.78	Lower Kittanning	Lycoming County	45.50
Upper Kittanning	Cambria County	76.13	Middle Kittanning	Mercer County	50.00
Lower Kittanning	Cambria County	77.81	Lower Kittanning	Mercer County	50.00
Clarion	Cambria County	68.78	Brookville	Mercer County	65.50
Brookville	Cambria County	68.43	Mercer	Mercer County	53.77
Lower Kittanning	Cameron County	60.27	Sharon	Mercer County	67.80
Brookville or Clarion	Cameron County	60.05	Alton	McKean County	50.05
Alton or Mercer	Cameron County	50.09	Clermont	McKean County	58.99
Upper Freeport	Centre County	94.68	Sewickley	Somerset County	80.50
Lower Freeport	Centre County	62.60	Pittsburgh Rider	Somerset County	83.00
Upper Kittanning	Centre County	Erroneous	Pittsburgh	Somerset County	89.40
Middle Kittanning	Centre County	55.40	Upper Freeport	Somerset County	61.45
Lower Kittanning	Centre County	72.40	Lower Freeport	Somerset County	59.72
Brookville	Centre County	49.28	Upper Kittanning	Somerset County	73.72
Upper Freeport	Clarion County	60.51	Lower Kittanning	Somerset County	64.10
Lower Freeport	Clarion County	60.47	Clarion	Somerset County	61.73
Upper Kittanning	Clarion County	52.74	Brookville	Somerset County	61.53
Middle Kittanning	Clarion County	53.44	Seymour	Tioga County	51.36
Lower Kittanning	Clarion County	69.13	Morgan	Tioga County	52.30
Upper Clarion	Clarion County	63.60	Bloss	Tioga County	94.65
Lower Clarion	Clarion County	66.26	Bear Creek	Tioga County	50.68
Brookville	Clarion County	61.97	Washington	Washington County	21.83
Mercer	Clarion County	53.00	Waynesburg	Washington County	34.94
Upper Freeport	Clearfield County	53.55	Redstone	Washington County	56.08
Lower Freeport	Clearfield County	68.30	Pittsburgh	Washington County	79.94
Upper Kittanning	Clearfield County	53.68	Freeport	Washington County	41.70
Middle Kittanning	Clearfield County	56.09	Waynesburg	Westmoreland County	53.12
Lower Kittanning	Clearfield County	63.25	Redstone	Westmoreland County	51.21
Brookville	Clearfield County	60.90	Pittsburgh	Westmoreland County	90.95
Middle Kittanning	Clinton County	50.00	Upper Freeport	Westmoreland County	69.66
Lower Kittanning	Clinton County	60.00	Middle Kittanning	Westmoreland County	42.15
Brookville	Clinton County	63.90	Lower Kittanning	Westmoreland County	59.33
Alton or Mercer	Clinton County	67.30			

[a] Authority: J. F. Reese and J. D. Sisler: Bituminous Coal Fields of Pennsylvania—Coal Resources. Topographic and Geologic Survey *Bull.* M-6, Part III (1928).

TABLE 4.—*Pennsylvania Anthracite*[a]

	Bed	Recovery,[b] Per Cent		Total Recovery,[b] Per Cent
		First Mining	Second Mining	
	NORTHERN FIELD			
K	Snake Island	43.2	14.6	57.8
J	Abbott	54.6	15.0	69.6
I	Top George	52.8	19.0	71.8
	Kidney	52.0	15.3	67.3
H	Hillman orchard	50.6	20.5	71.1
G	Top split	55.4	10.8	66.2
	Bottom split	54.3	24.0	78.3
F	Top split	47.8	21.0	68.8
	Bottom split	55.0	16.8	71.8
E	Top split	44.4	18.4	62.8
	Middle split	49.0	16.9	65.9
	Bottom split	45.7	12.8	58.5
	No split	42.7	16.1	58.8
D	Top split	50.1	28.0	78.1
	Bottom split	51.4	2.9?	54.3
	No split	53.3	14.9	68.2
C	Top split	53.0	18.2	71.2
	Middle split	45.7	16.0	61.7
	Bottom split	47.9	26.9	74.8
	No split	56.4	19.4	75.8
B	Top top split	50.6	23.9	74.5
	Bottom top split	56.4	28.8	85.2
	Middle split	53.4	25.8	79.2
	Bottom split	46.3	28.0	74.3
	No. 4 Dunmore	45.2	23.1	68.3
	No split	42.9	12.8	55.7
	EASTERN MIDDLE FIELD			
L	Little Tracy	45.0	40.0	85.0
K	Big Tracy	45.0	22.3	67.3
J	Upper Diamond	50.0	22.6	72.6
	No. 2	50.0	14.5	64.5
	No. 1	50.0	16.3	66.3
	Lower Diamond	50.0	20.9	70.9
I	Little Orchard	50.0	23.1	73.1
H	Orchard	50.0	15.8	65.8
G	Primrose	50.2	20.3	70.5
F	Holmes			
E	Mammoth	52.6	20.0	72.6
D	Wharton	48.3	26.3	74.6
C	Gamma	50.8	26.5	77.3
B	Buck Mountain Little Buck Mountain	54.2	21.4	75.6
A	Lykens Valley	53.9	27.1	81.0
	WESTERN MIDDLE FIELD			
K	No. 14 Tracy	40.0	19.0	59.0
J	No. 13 Diamond	36.9	29.8	66.7
H	No. 12 Orchard	36.5	29.1	65.6
G	No. 11½ Primrose	35.6	29.6	65.2
	No. 11 Primrose	37.5	30.0	67.5
F	No. 10½ Holmes	44.4	29.6	73.0
	No. 10 Holmes	37.7	22.7	60.4
E½		36.3	25.0	61.3
E	No. 9½ Mammoth	39.7	26.4	66.1
	No. 9 Mammoth	40.8	27.3	68.1
	No. 8 Mammoth	39.6	28.2	67.8
	No. 7½ Mammoth	36.1	25.4	61.5
D	No. 7 Skidmore	40.4	28.6	69.0
C	No. 6 Seven Foot	41.3	26.2	67.5
B	No. 5 Buck Mountain	36.7	27.4	64.1
	No. 4a Buck Mountain	32.9	15.4	48.3
	No. 4 Buck Mountain	39.0	25.5	64.5
A	No. 3 Lykens	48.9	44.1	93.0
	No. 2 Lykens	47.5	29.2	76.7
	No. 1 Lykens	48.0	25.0	73.0
	SOUTHERN FIELD			
N	Peach Mountain	41.8	29.2	71.2
L	Little Tracy	41.6	29.3	70.9
K	Four Foot	35.4	25.0	60.4
	Tracy	41.2	28.9	70.1
J	Little Diamond	33.6	24.0	57.6
	Diamond	37.8	27.0	64.8
H	Orchard	21.4	17.9	39.3
G	Primrose	29.0	25.0	54.0
F	Holmes	40.5	28.7	69.2
E½		45.0	40.0	85.0
E	Top Mammoth	38.7	29.9	68.6
	Middle Mammoth	34.6	30.7	61.3
	Bottom Mammoth	42.7	25.7	68.4
	Mammoth	25.7	22.4	48.1
D	Skidmore	39.2	27.1	66.3
C	Seven Foot	35.9	31.2	67.1
B	Buck Mountain	34.8	33.9	68.7
	Little Buck Mountain	48.7	0	48.7
A	Big vein	36.9	32.6	69.5
	Little vein	29.3	34.3	63.6

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[a] Authority: Dever C. Ashmead: Anthracite Losses and Reserves in Pennsylvania. Topographic and Geologic Survey *Bulletin* M-8-1926.

[b] All percentages of recovery are weighted averages.

Valuing Coal Properties

By H. N. Eavenson

In valuation of coal properties, no general rules can be laid down. Experience and probable future trend of the industry must be the deciding factor. No coal lands have any value that will not be mined within 40 years. Many more companies and individuals have lost money by having too many reserves extending beyond 40 years of life than by having too few. No fair relation exists between prices and present value of royalties computed for 40-year life, using Hoskold's formula and interest rates of 4 and 8 per cent. Where custom is to purchase lands, sales prices usually will be somewhat higher than values determined from royalties in this manner, but the condition is reversed where leases are customary.

A valuation for insurance purposes should be cost of reproduction at time of valuation, less depreciation, and the new plant should not necessarily be an exact duplicate of the old, but one having the same capacity.

Depreciation should be charged per ton and should be laid on the plant as a whole and not on each item. Salvage values are practically nil and should be neglected in determining a value. Rarely do their net returns exceed 5 per cent of the cost.

By assumptions of cost of plant, output, life and earnings per ton, the most economical life of the coal property can be determined, but a change in any of the variables will affect the result materially, and it is unlikely that any coal mined was ever worked out on this basis. Increasing cost of machinery and housing for modern mines tends to increase their life.

Excess reserves, beyond a reasonable life, should be valued as coal to be developed after that period.

Only cost of yardage or improvement of headings in excess of that needed for normal mining should be charged to development account. Until a mine has reached its designed output, all mining should be charged to development, and that item should be debited with all coal sold. After that, all rails, etc. should be charged to mining, and only expense of unusual work, such as driving through rolls, grading headings, etc., to development, and this charge should be depreciated in proportion to tonnage mined.

Nothing less than 6 per cent, or better 7 per cent, should be used for any royalties, and 8 to 10 per cent for many of them. For operating owners, rates should be 10 to 15 per cent and for lessees from 10 to 20 per cent, the former figure being used rarely and only in exceptional circumstances.

Value of property to the lessor is the present value of the expected royalties at rates of interest selected; to the lessee it is the difference between total value and that figure.

Value of Coal Land as Restricted by Economic Life

By G. W. Evans

When a mine has been developed by a low-cost plant and to produce a medium tonnage, and the coal is held at a fairly high value in the ground, its normal life might well be taken at from 15 to 20 years. Coal in excess of this life should be graded downward in such a manner that values will cease in, say, 45 or 50 years. The normal life of a mine with an elaborate and expensive plant and development, and with coal of lower valuation, might be extended to, say, 40 years, and the coal carried on the books at a reasonable valuation for another 20 or 30 years.

To determine the value of the coal in excess of what may be mined in the normal life of the mine, definite values must be apportioned to the coal, annual per-ton cost and prospective earnings. After a determination of these variables has been made, curves can be developed that will give the answer desired. Except for specific examples, the question—What is the correct life for a mine?—cannot be answered.

Mine development should be continued until it reaches the output projected at the start of the project. If, for instance, the plans made call for an expenditure of $300,000 for mine development to bring the property to a capacity of 3000 tons a day, and owing to unforeseen obstacles it is found necessary to expend another $150,000 to bring the mine up to the required tonnage, the additional $150,000 should be charged against the value of the property.

Until a mine is able to produce the projected capacity, development should be charged to capital cost. The actual cost of shafts, slopes, drifts, gangways, rock tunnels, etc., should be charged probably as class A. Minor openings might be given class B rating. The entire development costs should be charged on the books during that period, less the estimated normal cost of the production per ton. The value of any development during the life of the mine is either the actual cost, or (if the books are not available) an estimate of just what this work would cost based on the practice of the district. Most of this can be estimated within reasonable limits.

Depreciation should be heavier at the early stages of the mine and taper off toward the end to compensate for longer hauls and greater cost of ventilation.

To the lessor or owner, the value of coal in the ground will probably be determined by the current values for the district in which the property is located. If he is a good trader, he might set his values high, but if he is honest with himself, he will be conservative, so as to induce someone to risk his money in developing the property. He should be satisfied with one-fifth of the expected net returns on the coal to be mined, for he takes little or no risk, and if a coal mine is developed, the value of his property is enhanced.

The yardsticks some mining engineers use to evaluate coal properties —tonnage of coal in the ground and some arbitrary value, say five or ten cents a ton—in my judgment are unreasonable. Dozens of instances could be cited where tonnages can be calculated within reasonable bounds, and where the quality of the coal is proved to be as good as the average in the district; yet to value this coal at even a dollar an acre and pay taxes on it would be financial suicide. I have even advised a large western company to return to the government coal lands containing hundreds of millions of 20 to 30-ft. coal, because the company had in other areas enough coal to satisfy present-day market requirements for 800 years. Their descendants can more profitably pay a nominal royalty to the government at that time than carry the burden even at a very low valuation. Many men, including tax commissioners and assessors, delude themselves into believing that all coal land has some value. In hundreds of instances it is a liability rather than an asset. Why should not coal be regarded as other minerals of low value—clay for brick, trap rock for roads, limestone for cement, gravel for road and building purposes? Why should those who own coal lands be singled out to carry the taxes of the communities where coal lands exist? Let the local assessor give a reasonable valuation to the coal that will be mined in intervals of 10 years for, say up to 40 years, and value the rest as wild land.

Companies with coal lands on their books at so much per acre should try to borrow money on it, and they would soon find out what the property really is worth.

Value of a Developed Coal Property Apart from Its Appurtenant Coal Lands

By John A. Garcia

If a coal-mining plant is to be valued separately from its coal lands, the value should be based on replacement costs using present market prices for labor, material, equipment, etc. This replacement cost should be reduced by the sum necessary to put the plant in a condition as good as new, and at the same time due consideration should be given to the remaining life of the mine. Even if the plant were new and practically no expense were necessary to make it "good as new," the present value would be practically nothing if the mine had only a year or so of life. Though an extreme illustration, this shows that available acreage or life is an important factor in determining the value of a mining plant. Original cost should be given no consideration. The cost of a plant constructed in time of war has no relation to present-day value. There may be cases where original cost must be determined for income taxes, etc., but that consideration has nothing to do with present-day values.

When a mining town is so located that it has earning power apart from the mine and has houses rented to others than employees, it should be valued separately from the rest of the plant. However, it is difficult to imagine a mining town independent of a mine or mines in the vicinity, and if the prosperity of the town depends on mines other than the one to be valued, the life of the other properties must be determined. Again, the type of houses must be considered, and replacement value depreciated at a rate consistent with the type of structure.

Salvage values of a coal-mining plant bear little relation to first or replacement costs. Hoists, locomotives, cutting machines, etc. soon have little value because of new and more efficient design. The breast-type of cutting machine soon became valueless because of the introduction of the shortwall cutter, and now it appears that the track-mounted cutter is about to supersede the latter. From lack of standardization of the equipment at coal mines, equipment at one mine rarely can be used at another, because track gage, over-all height, width, etc. vary so much from mine to mine. Hoists also must vary because of shaft depth, gross hoisting load and other vital factors. Little, if any, can be saved by dismantling and re-erecting wood or steel structures, all factors considered. Salvage value in general bears no ratio whatsoever to new cost, and the equipment and structures of an abandoned coal mine can be disposed of only through junk dealers, advertising and bargaining.

Evaluation of Coal Properties

By J. A. Grimes

Coal valuation may be based on: (1) actual sales prices of similar coal, (2) royalty received for similar coal, (3) estimated earnings from operation, but no one of the three methods is generally applicable and each has its field of use.

For valuing undeveloped coal lands, where there are a fair number of actual sales of similar lands at essentially the same prices, at or about the date of valuation, actual sales prices of similar acreage would probably be the best measure of value. But indiscriminate use of this method may lead to absurd conclusions. Suppose, for example, during the peak of the war demand for coal, 20 acres were sold for $500 per acre; the buyer might expect to exhaust this acreage before the war demand subsided, and $500 per acre would be a fair market price for the tract purchased because it would yield him handsome profits. If 1000 acres of similar coal had been offered to the same operator at $200 per acre, he might well have refused it, believing it could result only in loss.

Royalty received for similar coal likewise may give absurd results with indiscriminate use. When lessor and lessee agree on a royalty rate, the royalty comprises only the value of the coal in place, if the coal acreage is undeveloped and unequipped. If it is developed and perhaps equipped, the royalty combines payment for the coal and rental for development and equipment. As royalty for coal and royalty for development and equipment are seldom separately specified, royalty rates on developed coal properties are inapplicable for the valuation of undeveloped or differently developed properties, and vice versa. On the other hand, royalty rate at leasing date is but a fraction of expected profits from operation. The other fraction is conceded to the lessee operator in return for his risk and for the management and capital which the lessee supplies. Royalty on coal comprises the entire value of the coal, plus interest on the deferred royalty payments. The principal difficulty in valuation by the royalty method is to separate royalty payment into capital and interest. The interest which a lessor demands will be governed by many considerations, including type of operation lessee proposes, financial responsibility of lessee, rate of production, percentage recovery of coal, ownership of plant improvements by lessee or lessor, should lease be forfeited, etc. Obviously, a lessee might well be satisfied with lower royalty from a thoroughly responsible lessor than from one with less or undemonstrated responsibility. Thus, royalty rate for the same property at the same date may vary according to risk of future royalty receipt, although the cash price for the property would be the same to either lessee in the event that the two lessees should become prospective buyers for the property.

Increase of royalty rates for a period of time is a fair measure of appreciation in value of a lessee equity during that time. Valuation of royalty increments must be subjected to the same exercise of judgment as the valuation of the lessor's expected future royalties.

Valuation of estimated earnings sometimes is the only possible basis for valuation, particularly of going concerns as distinguished from undeveloped coal properties. This valuation from an undeveloped property should be identical with that determined for expected royalty from the same property, should the owner have the option of either operating or leasing the property at date of valuation. Valuation of royalties and valuation of operating profits from any coal property can fail only to yield the same results at the same date if the various estimates made and factors considered in the two valuations are erroneous. The proper use of either method depends entirely on the judgment, knowledge and experience of the evaluator, and not on any mathematical formula that may enter into the calculation. Accuracy of any valuation depends more on the evaluator than on the method employed. Some have more experience with one method than another and invariably get more accurate results with the method with which they are most familiar.

With equally accurate valuation factors available for all three valuation methods, and with an evaluator equally competent to use any of the three methods, the comparative sales method would be first, the royalty bases second, and the operating profits valuation third, in obtaining accuracy and ease of application.

Surface-property valuation depends also on the judgment of the evaluation. When mining will destroy or deteriorate the surface, the latter has little or no value aside from the coal, and may even be a liability, as local taxes may have to be paid far in excess of any possible future sales value of the land after the mine is exhausted. When the value of the surface is not destroyed but the ground is likely to subside, a prudent owner will not sell the surface, subject to the risk of damage suits, until mining operations are concluded. In such a case, the surface value would be present worth of estimated sales price of surface at conclusion of mining operations, less present worth of taxes and other carrying charges during operation.

When surface and mineral rights can be severed without detriment to mining and without danger of damage suits, surface rights should be valued at current sales prices. If land for well located surface plants is scarce, it may have actual sales value above the value of equally good agricultural land, but when the coal is exhausted, this excess value has disappeared completely and, therefore, should not be regarded as of value apart from the coal deposit. Cost of acquiring an expensive outlet site should be treated as part of the cost of coal.

An evaluator should be conservative in assigning value to surface separate from coal, as these values are usually based on predictions of sales value at a distant future date, or are subject to reduction by the cost of leaving heavier supporting pillars or the cost of future subsidence damages.

Two otherwise similar coal tracts will have a difference in value equal to the estimated difference in cost of future development.

The risks of evaluation with inadequate prospecting will vary with the irregularity of the coal seams, as demonstrated by other operations in any given field. If chances of irregularity are great, or if small differences in thickness or quality will destroy the value of the coal, the risks are great. Experience in evaluation of metal mines has shown that when profit estimates have been demonstrated to be accurate by actual operating results on a commercial scale, the stock market value of the property will be about double what it was before engineers' profit estimates were confirmed by operation. Value of coal acreage, proved by development and actual mining, should be about twice that of unproved or partly proved acreage which is apparently similar in other respects.

Whether coal and surface should be valued separately from plant and mine depends on nature of ownership largely and also on the property being valued. If either plant or development will have a salvage value after coal is exhausted, the present worth of such salvage value should be considered in any valuation. In England, more than in the United States, payments are collected for use of entries of exhausted higher mines by the owners of underlying coal. With relatively flat and shallow coal seams, such salvage values become negligible. If the mine owner is a lessee, having right to remove mining plant when coal is exhausted, the salvage value of such plants may be a small element of value to the operator. If the lessor becomes the owner of mining plants at termination of mining, this value belongs to the lessor.

Going concerns are usually valued more by their profits and dividends than by any other basis. It is usually impracticable, if not impossible, to separate the value of profits among its various sources—coal, surface, Slant, development, organization, management and sales contracts. pales contracts which provide a regular year-round market for one mine, while a neighboring and similar property can operate only seasonally, and successful management are items that swell profits, but it is difficult to assign definite values to them. Going-concern value must be determined generally *in toto*, without division into separate elements of value, loss of any one of which might destroy the entire going-concern value.

A plant must usually be valued separately from other parts of a mine for insurance purposes. Customary basis for insurance and for coal mines owned by public utilities is cost of reproduction less depreciation. When a plant is valued to ascertain cost of depreciation chargeable to operations, the original cost, less depreciation, should be taken. This basis must also be taken for Federal income-tax purposes, except that values of properties on March 1, 1913, may be substituted for cost, if the property was owned before that date. Actual cost is the most satisfactory basis for determining prepaid operating charges for depreciation, though that position is a subject of much controversy. The average appraisal determines reproductive cost with reasonable accuracy, but no engineer can determine extent of depreciation or obsolescence by visual examination.

If one would not replace the plant in use with an identical plant, but would build an improved plant if the existing one were destroyed, why should the cost of reproducing the plant ever be considered? If reproductive cost of existing plant gives a depreciation charge of 2¢ more per ton of annual production than would result should depreciation be computed on original cost, but use of a modern plant would reduce operating costs by an equal amount, why should one add 2¢ per ton to operating charges over the cost of operation with a modern plant by computing depreciation on the reproductive cost of the old plant. Appraisals have great value, especially to companies with inadequate property records, but they should not be used for cost accounting.

Towns are separate properties and should be valued separately from the mining property.

Climatic conditions, mine water and other factors vary so materially in different localities that the determination of proper depreciation rates for coal plants is a local matter. Depreciation rates per ton for entire plant and equipment are preferable to rates for individual items, or yearly fixed percentage rates for items or entire plants. The per-ton rate for an entire plant can be determined only with reasonable accuracy by estimating the total cost of the plant and equipment required to exhaust the mine and the division of this total estimated cost by the total tonnage that will be mined. Depreciation of prepaid development costs would be computed similarly. With this method, periodic re-estimates of total plant cost and tonnages are necessary, and depreciation per ton must be adjusted to these re-estimates as obsolete plants will be replaced from time to time.

In general, salvage values are so small, after costs of salvaging are deducted, that they may be ignored. Small movable equipment, such as mine locomotives, cars and mining machines, may have important salvage values, but those of long-lived and heavy equipment are negligible. When the Anaconda smelter was moved three miles across a valley on a company-owned railroad, though it was well equipped and not obsolete, its salvage value was barely 5 per cent.

Operating profits more than 20 years in the future are rarely considered by prospective buyers when values are based on future expected profits. Some engineers use an arbitrary life of 30 years, and for Federal income-tax purposes, an arbitrary maximum life of 40 years was used in valuing expected future profits from metal mines.

Excess property that will not be exhausted within the normal life of a mine is rather a liability than an asset, as carrying charges are practically certain to be greater than cost of buying reserves in the same or another field when coal supply for normal life has been exhausted.

As any remaining acreage in excess of that needed for the normal life of a mine might be available for sale or development by new mines, such coal should have some value, but in general this would involve a change of policy, which is by no means probable and certainly not assured. Carrying charges of excess acreage might be charged against the profits of operating mines and the valuation made on an acreage basis at current sales prices. If the operating profits on which the value of the coal land is based depends on the elimination of competition of other mines, this value might be greatly impaired by the sales of such excess lands. Even though the unused lands be sold to a railroad or metallurgical company for immediate use, the value of the operating company's property might be decreased, in case it thereby lost the opportunity to sell its product to such consumers. Sometimes a railroad or metallurgical company may retain coal lands in excess of those needed for the normal life of its mines to protect its investments in trackage or other fixed, non-coal-producing property.

Any mine with coal developed ahead of extraction has a capital investment in the prepaid operating cost of development. The total cost of development required to exhaust the property should be estimated, and this should be divided by the total tonnage of minable coal to get development cost per ton. This cost should be multiplied by tons mined in the past to ascertain cost of consumed development, and this in turn should be deducted from past actual or estimated expenditure for development to ascertain value of existing development in advance of coal extraction.

Costs of operation—haulage, pumping and ventilation—increase as a mine grows deeper and more extensive, so it would seem not unfair to charge costs of development in earlier years against the coal obtained, even if some of the development is in rock. But sometimes the roadways in early development are made larger and have more permanent support, which argues against charging such excess expenditure to current coal cost. Large book values for development arise largely from evaluation by accountants, made without the advice of engineers and also from the fact that so few engineers care to develop such a knowledge of accounting problems as would give value to such advice as they might give.

Different interest rates on capitalized earnings or royalties should be assumed for lessors, operating owners and lessees. Highest rates should be given the lessee, less high rates the operating owner, and the lowest rate should be given the lessor, as the lessor has less risk and the lessee more risk than the operating owner. Proper differentials in such rates may be ascertained or estimated most accurately by considering royalties and expected operating profits when the leases are made. When a lease is contracted the lessor gives nothing to the lessee except a prospect of making a certain operating profit in return for risks assumed and services rendered by the lessee. At the time the lease is made, therefore, the interest rates used for discounting expected future royalties, and expected future operating profits to present worth, should be such that identical values will be ascertained by either method of valuation. In Lake Superior iron-ore leases, the lessor gets 40 to 67 per cent of the expected future profits, and the lessee what remains, with 50 per cent to each a rather customary division. Probably, somewhat similar ratios could be ascertained for coal valuation. Then, valuation could be based on the deduction of that percentage of the estimated operating profits for risk and services, including cost of development and construction, with the remaining operating profits valued as royalties from the coal itself.

Rate of interest in valuation of coal royalties would depend on certainty as to rate of future production, regularity with which future royalty payments may be anticipated, financial responsibility of loss, etc. For fixed minimum royalties, the lowest rate of interest would obtain—one almost identical with the prevailing local rate for well secured first mortgages on real estate. For royalties in excess of guaranteed minimums, higher interest rates should be used in discounting to present values, at least 8 to 10 per cent as compared with 5 to 6 per cent for guaranteed minimum royalties. For operating owners, 10 per cent would be a minimum rate, and for lessee equities 12 or 15 per cent. The maximum rates for operating owners and lessees might be much higher, but 20 to 25 per cent would be none too high for lessee equities in a fairly large part of all valuations. With centuries of experience the more competent English evaluators in their published works on valuation use the Hoskold formula and rates about the same as those stated, though American writers in general use lower figures, which are neither well considered nor in line with actual sales prices for coal properties.

The greater the margin of profit in the operation of the property, the lower the risk, as such a property can operate in times of depression without loss or at reduced profits, while its competitors are going bankrupt. Another element of risk is mounting rate of local taxation, which if disproportionate in one region, and if not as burdensome on rival fuels, may make it impossible for the coal operation to absorb such charges.

Theoretically, the potential rate of production should be used instead of the present rate in estimating income and life in a valuation based on capitalized earnings or royalties, but in practice the prices paid for going businesses are more generally based on past performance under competitive conditions than to any advantages a business may gain over competitors. Estimated profits from future predicted increases in production will, therefore, be valued generally as subject to much greater risks of realization than profits which past performance has demonstrated as entirely probable.

An operator's working capital should provide, not only for the production, storage and transportation of his coal for the period of time required to get the coal from the mine to market and to collect payment for it, but also for meeting the risks of the business. Ratio of working capital to expenses must vary greatly with regularity or seasonality of production, the type of customers supplied and the sales contracts in force.

A property has only one actual or market value at any time. Its assessed value may be only a fraction of actual value, and bankers may seek to inflate the actual value if they are selling security issues, or to deflate the true value if they are lending their own funds. The true value is that which would be fairly fixed by a willing buyer not under compulsion to buy and a willing seller not under compulsion to sell, where both are fully informed as to all pertinent facts and both have equal intelligence and judgment. Evaluation of a property is merely the approximation to the sales price which should obtain under theoretically ideal circumstances. Actual sales price may vary from true value. Purchase and resale at profit or loss, almost simultaneously, is too common to leave this statement in question.

Though value of a going concern varies with variation in operating and marketing ability of the operator, the real value of the property does not thus vary.

Profitable Life and Value of a Coal Mine

By Otto Herres

Normal profitable life of a coal mine is the operating period during which invested money will bring maximum returns. Factors on which such returns rest are market or sales possibilities, geographic location, accessibility, physical difficulties of development and operation, transportation, facilities, labor conditions and supply, sources of material and supplies, coal reserves, thickness and impurity of seams, gas, water, pitch, regularity, roof and other geological conditions, and rank, analysis and physical characteristics of coal. To these must be added cost of plant and equipment and working capital, for operations and taxation.

The Engineers' Advisory Committee of the United States Coal Commission in 1924 stated that:

> Various methods of valuation of mining properties have been suggested, but all authorities seem to agree in accepting, as the fairest, and most practicable and logical, a capitalization of the estimated future earnings, when such estimate can reasonably be made. Hoover and Finlay both emphasize this.

It quoted Ingalls' "Wealth and Income of the American People," page 97, as saying:

> The fundamental principle of mine valuation is the present worth of an annual dividend accruing during the period of years corresponding with the life of the mine. It is a convention that the estimated period shall not be more than 30 years.

Value of a coal mine depends on how much the mine can earn and how quickly earnings are acquired. Property to the owner has only one value, present worth of future profit. But earnings vary with managements and with fluctuations of price and demand. A property at any given time may have but one true value, but differences of opinion regarding surrounding conditions may affect its price.

Assessed valuation for taxation is rarely a reliable indication of value, for assessment values vary widely in different counties and states and are likely to be based entirely on acreage, seam thickness and investment in plant and equipment, which are only a few of the factors entering into a true appraisal, but even physical features may not be given weight in proportion to their bearing on earnings.

Bankers' valuation for financing purposes may be a useful factor in determining existing market value, but such values may be inflated for the negotiations of a loan or a stock issue, and in times of depression such value may be written down below intrinsic worth.

Estimation of Coal Land Values

BY R. D. REEDER

VALUES of coal land should be estimated in accord with the decision of the United States Supreme Court in the case of the Cleveland, Chicago and St. Louis Ry. *vs.* Bachus (154 U. S. 439):

> But the value of the property results from the use to which it is put and varies with the profitableness of that use, present and prospective, actual and anticipated. There is no pecuniary value outside of that which results from such use. The amount and profitable character of such use determines the value.

In its report the Committee on Federal Taxation of the American Institute of Mining and Metallurgical Engineers appointed in 1919 says:

> A proper value of a mining property is the present value of the prospective net earnings taking into account probable variations in output and value.

Sales of coal and surface should be valued separately because: (1) Sales of mineral deposits are often independent of the surface and vice versa; (2) changes in value of surface occur owing to increase or decrease of its use; (3) changes in value of coal deposit occur owing to change in (*a*) market demand for the product, (*b*) cost of production, (*c*) management.

Accessibility is the most important controlling factor in the value of coal land, for transportation is one of the largest single items in cost of coal to the consumer. In mountainous regions, fee value includes coal, surface and timber and the last must be considered in valuing the property. Commercial timber and surface must be valued separately; mine timber is merely incidental. In some cases, the surface may be of so little value that it need not be considered. In parts of the United States, royalty rate and selling price may bear no relation; the royalty rate may be 10¢ a ton and selling price $20 an acre.

If the plant is in operation, it should be considered as an industrial unit and valued as a whole. Frequently land and buildings are indistinguishably merged into a single account. In such cases, the two items should be valued separately and thereafter kept in separate accounts. Land values include examinations, costs, title, registration agent's commission or salary, taxes accrued to date of title as well as any liens on title when these are assumed by purchaser; also cost of assessment for public improvements which enhance land value, cost of grading not properly chargeable to building and expenses of condemnation proceedings. Where a large acreage is involved, as 100,000 acres, and only 12,000 or 16,000 acres can be mined in 40 years, the remainder should be valued by the acre and not as part of the industrial unit.

Rates of interest on capitalized earnings or royalties where the valuation is based on one or other should recognize the following facts: To insure a profit a buyer demands a margin on estimated tonnage value to protect him from: (1) risk of investment, (2) delays in mining the coal, and (3) overestimate of recoverable tonnage. Risk rate and expected profit must cover: (1) variations in price of coal at mine, (2) hazards due to competition and overproduction, (3) hazards due to change in natural conditions, (*a*) bad top, (*b*) bad bottom, (*c*) irregular gradients in coal seam, (*d*) faults and other hazards beyond control of operation, as explosions, fires and strikes.

For example, if the margin is fixed at one-half estimated value, coal should have a sales value of 2½¢ per ton, if royalty is 10¢ per ton. Legal interest rates in the state have no bearing on the rate of interest applicable, as risk involved is the most important item to be considered. Minimum interest rate advisable should be 8 per cent; the maximum is dependent on the apparent earning capacity. In many cases, rates have exceeded 20 per cent.

Maximum interest rate or risk rate will depend on: (1) inability to forecast mining conditions, (2) inability to forecast costs beyond a short period, (3) possibility of losing market from (*a*) use of oil, (*b*) use of gas, (*c*) use of other coal, (*d*) change in freight rate, (*e*) change in smoke laws. Profit may be considered to be in inverse ratio to cost of mining. Initial investment should invariably be refunded within first 20 years of life of mine.

Ultimate Cost of Coal per Dollar of Purchase Price

Amortization period, 40 years	
Rate of interest, 5 per cent	$0.05
Annual taxes, 1 per cent	0.01
Annual amortization charges	0.03
Total annual cost	0.09
Multiply by amortization period	
Total cost per dollar of purchase price at end of amortization	$3.60

Income and life should not be based either on actual or potential rate of production, but analysis should be made of market conditions, mining methods and rated capacity of plant. Workability of a coal deposit depends on: (*a*) character and heat-giving capacity, (*b*) accessibility, quantity, thickness, depth of deposit and other conditions affecting production cost.

An example will clarify my statements. A coal property of 2000 acres, owned in fee simple (surface and coal) originally underlain by a 5-ft. level coal seam, containing a mine and plant 10 years old with normal production (past and prospective) of 240,000 tons a year; today's cost of installing such a mining plant (buildings, structures, equipment, development and town), $300,000; estimated profit after all direct operating charges but before such overhead expenses as depletion, depreciation, federal taxes, and interest on investment, 30¢ per ton. Assuming or deriving such other factors as may be necessary, what is present total value of such a property and how would this total be divided among major components, unmined coal, surface, plant and town? Also, what would have been a proper royalty for the original owner of the coal to have demanded, had he leased it to present owner in its virgin state?

Coal yield per acre (about 85 per cent recovery)	7,500 tons
Total recoverable coal	15,000,000 tons
Average annual production	240,000 tons
Estimated life of mine	63 years
Normal life of mine	40 years
Total coal to be mined in 40 years	9,600,000 tons
Coal unmined at end of 40 years	5,400,000 tons
Average gross operating profit per ton	30 cents
Total gross operating profit per year	$72,000

Assuming plant replacements to be made out of annual earnings at the rate of $12,000 per year:

Estimated profit, 40 years	$2,880,000
Original cost, plant and equipment	300,000
Present value (depreciated cost)	180,000
Remaining life of plant and equipment, with entire life of both 25 years	15 years
Average cost per year to replace and maintain plant (depreciation)	$ 12,000
Total earnings after plant replacement	$2,400,000
Present worth (4 per cent sinking fund and 12 per cent interest in 40 years, Hoskold's formula)	$ 459,600
Value present plant	$ 180,000
Value of coal to be operated in 40 years	$ 279,600
Value per acre	$ 218.44
Value per ton	$ 0.02912
Coal remaining after 40 years, tons	5,400,000
Profit after 40 years at 30¢ per ton	$1,620,000
Life after 40 years	23 years
Plant replacement (20 × $12,000)	$ 240,000
Net earnings ($1,620,000 − $240,000)	$1,380,000
Present worth (20 years, 4 and 12 per cent)	$ 449,328
Value of surface @ $5 per acre	$ 10,000
Total value	$ 459,328
Present value @ 6 per cent compound interest	$ 43,646.68
Value per acre (720 acres unmined at end of 40 years, $43,646.68 ÷ 720)	$ 60.62

The following method is suggested:

Total expected income			$2,880,000
Present worth (40 years 4 and 12 per cent)			$ 540,259.20
Life present plant	15 years		
Life of new plant	25 years		
Total	40 years		
Cost of new plant	$300,000		
Present worth (4 per cent, 40 years)		$148,444.20	
Value of present plant		$180,000.00	$ 328,444.20
Value of coal			$ 211,815

In the foregoing solution, a normal plant is assumed to cost $300,000, and that plant and equipment will have an average life of 25 years, the cost per year being, therefore, $12,000, which would also be the depreciation per year. If it is assumed that no amount will be set aside each year to provide for additions and replacements to plant and equipment, payments for these must be made out of earnings at an average rate of $12,000 yearly. This reduces expectancy of income to $2,400,000 ($2,880,000 – [12,000 × 40]).

In arriving at the value of the coal remaining after 40 years, it may be assumed that the original plant, by reason of the additions and replacements, is in good condition at the end of the first 40 years. In this case an average of $12,000 may have to be expended per year for 20 out of 23 years of its remaining life, though perhaps in the last 5 or 6 years no additions other than ordinary repairs and supplies would be needed, such as ordinarily constitute a part of normal operating cost.

Assuming it will be necessary to spend $12,000 for 20 years, the value 40 years from now would be $449,328. Hoskold's formula will not apply because there is no income for 40 years. The value of $449,328 today, at 6 per cent compound interest, is $43,646.68 including surface.

In calculating a suitable royalty rate, the value per ton as shown is $0.02912, which represents present worth of certain earnings per ton at 4 and 12 per cent for 40 years. The factor for this, according to Hoskold, is 0.1915, which is the factor per dollar. Earnings per ton, the present worth of which is $0.02912, are $\frac{0.02912}{0.1915} = 15¢$. In other words, 15¢ royalty per ton payable as coal is mined is equivalent to a cash purchase price of $0.02912 per ton. It is doubtful whether royalty rates are ever fixed on so scientific a calculation. The nearest approach to a scientific royalty is based on a certain percentage of the selling price of coal, which, of course, has no relation to the operator's earnings.

Comparative Value of Undeveloped Coal Lands with Other Lands Adjacent

By S. A. Taylor

Where a coal property is in an undeveloped field, its value may be based on the sales prices of, and royalties paid on, similar coal properties, if the development cost in both cases are the same and if both would serve the same market; but due weight must be given to accessibility to means of transportation and to the effectiveness of the service of transportation after the property is developed. Fair allowance must be made for the relative level of the coal and the surface, and whether a drift, slope or shaft would have to be made to reach the coal. Pitch and gradients and ease of drainage and ventilation also affect values, and the appraiser must use judgment in comparing the property being evaluated with other properties of which actual sales, prices and royalties are known. Seam thicknesses and analyses of seam also affect values. However, it would seem that the evaluation of land should be based largely on estimated earnings from operation rather than on sales or royalty values. The appraiser may have to resort to all three methods of evaluation if he would arrive at the fair value of the coal.

If the surface to be valued contains more land than would be needed for mining town, mine buildings, railroad sidings, etc., the excess land should be given only the fair market value of similar land in the community. Land suitable and necessary for the plant site should receive a valuation appropriate to its suitability for that purpose. In a mountainous country, flat land may be worth many times as much as hillside property. Some measure of this value could be determined by a consideration of the difference in cost of construction of buildings and necessary adjuncts to the mine. A large portion of this difference in construction cost could properly be allowed to the land for plant purposes. The appraiser should remember that the real value of any coal property is the profit that can be derived from it by its operation; so the value must always bear a fair relation to the cost of the plant improvements needed to make it a going mining property.

The appraiser should also endeavor to discover any special advantages in coal land, such as availability of labor to the property, special markets for the coal at low cost of transportation, whether the property has available both railroad and water transportation to market, or whether two or more railroads are available for such transportation; also the character of the service rendered by such railroads and the markets reached by them.

Methods of Valuing Coal Properties

BY BLEECKER L. WHEELER

IN examining and evaluating coal and other mineral properties the most logical and satisfactory method is to determine a separate value for each of the several classes of assets of which the property consists. These individual items ordinarily may be classed as:

1. Physical property (buildings, machinery, equipment, etc.),
2. Land (surface only),
3. Mining developments (shafts, tunnels, stripping, etc.),
4. Coal (or other mineral) deposits,
5. Intangibles (good-will, going concern).

As the present value of the property is to be determined, it is self-evident that reproduction cost is the factor and that original (or so-called historical) cost is not of particular significance. But even this reproduction cost, less accrued depreciation and obsolescence, may be subject to certain important exceptions if, for instance, conditions exist that make it clearly impossible to operate the whole property profitably. Under such circumstances, reproduction cost naturally would not be the governing factor, as the physical property (having no earning power) would have only salvage value.

The same reasoning applies to the surface land and to the existing mining developments on the property. Surface land is worth what it will bring in the market without regard to original cost, though this value may be difficult of determination where no comparable land has recently been sold. Similarly, mining development is to be valued primarily on its reproduction cost, less deterioration, but all such work may not rightly be included in the valuation, for much of it may be of no remaining value to the property because of partial exhaustion of the coal or numerous other related conditions.

A method sometimes used in valuing coal deposits (if it can be dignified by the term "method") is to assign a "per-ton value" to coal in the ground and multiply this unit value by the estimated tonnage of recoverable coal, the assigned value per ton being usually established after consideration of recent sales of other coal properties, prevailing royalty prices on leased coal, etc. No matter how carefully this per-ton value may be determined by any such computation, no consideration is given to the "present value" of future earnings from the sales of the coal. If a deposit of 1,000,000 tons of recoverable coal is assigned a unit value of 3¢ per ton, thus being worth $30,000, it is clear that 2,000,000 tons of this coal will not necessarily be worth $60,000, for the second million tons will not be mined and sold during the same period of time as the first, and, consequently, will have far less value. A proper discount factor will have to be applied to arrive at its present worth. Frequently also, little consideration is given to the all-important factor—whether the coal be profitably mined—although any real value depends on profitable operation.

The normal profitable life of a coal mine depends on the workability and sufficiency of the coal deposit, future wage scales, costs of material, freight rates and other items of expenditure which affect the estimated cost at which coal can be laid in the market in future years, as also on continuance of demand, competition from other coal operations and from other fuels. When the profit margin is permanently reduced below a reasonable minimum, the "profitable life" reaches an end.

If more coal is contained in the property than can be handled during the normal life of a mine, it must be worth at least what the deposit could be sold for in open market. Adjacent mining properties may be able to utilize this excess coal, but sale of such coal to a competitor might be inadvisable because it might decrease current sales price of the product. Thus its value unused might be somewhat greater than the best sale price obtainable. Each case depends on local conditions and special factors, but the value of this additional coal must be estimated on an essentially different basis from the value of coal that it is expected may be profitably mined.

Rates of interest on capitalized earnings or royalties cannot be established for all properties at all times, because conditions vary greatly. A relatively small coal property in Wyoming, with a somewhat narrow market and heavy freight rates, should be valued on the basis of an entirely different earning rate from that used for a large property of low-cost "strip coal" in Illinois or Indiana. General industrial conditions and the state of the coal industry, locally and as a whole, have a direct bearing also on the selection of the interest rate. Thus, the estimated value of a coal property in a depression may be much below the value some few years earlier. Speaking generally, the value of a coal deposit under depressed and severely competitive conditions, using the basis of capitalized earnings, would probably lie between about 12.5 and 25 per cent, depending on conditions. To use only one interest rate in computing a value based on capitalization of earnings—at least when the rate selected is greater than about 8 or 10 per cent—is a wholly erroneous procedure. A property value determined by capitalizing earnings over a stipulated period presupposes that the estimated capital will be recovered intact by the end of the period, and that annual net earnings will be obtained also at the specific rate of interest. If only a single interest rate be used in the computation, the redemption of capital is assumed to be accomplished at the same rate of interest as that provided for earnings, which in practice may often be unreasonable and impossible. For instance, where risks and other conditions are such that a rate of, say, 16 per cent is considered proper for earnings, it will probably be unlikely that the necessary redemption of capital can be accomplished at an equal interest rate. Hoskold's formula provides for this condition, and valuation tables have been calculated from that formula. These provide for a direct computation of value on the basis of two rates of interest. Any estimated value for coal deposits should be based on a computation of this nature.

Real value of any property at any specific time depends on future earnings and future life. Past earnings are not a direct factor in the determination of such value, except as they tend to indicate subsequent or potential earnings. Estimated earnings in the years ahead and not years past are those to be capitalized in determination of potential value. Thus the income and life of a property should be based on the potential rate of production.

Plant Capacity and Carrying Charges, and the Effect Thereof on Value of Coal Properties

By W. H. Craigue,* Washington, D. C.

(New York Meeting, February, 1933)

The scope of this article covers only the mathematical principles involved in discounting to present worth future expected profits and it is believed answers definitely two questions, which were raised at a recent meeting of the Committee on Methods of Valuing Coal Properties, as follows:

1. The accumulative effect of carrying charges on the value of coal mines showing definitely the number of years beyond which the coal may not be mined at a profit and subsequent to which the possession of any unmined coal will be a liability rather than an asset.

2. The effect on the valuation of coal mines, in the presence of the carrying charges described in question 1, of an increase in plant capacity and production.

In general, in considering these questions, it may be stated that the consideration of an increase in plant capacity and production follows as a matter of course after a determination has been made of the year beyond which no profits are to be realized (giving the carrying charges accumulative effect, or in other words capitalizing the carrying charges, as shown hereinafter). For example, if it is determined that any coal that is unmined after 45 years will have no value or less than no value, the question immediately arises as to whether or not a plant should be constructed that would result in a yearly production great enough to extract the entire tonnage in a time less than 45 years, and if so the number of years less than 45 years that would, from a financial standpoint, be the most desirable, all under the assumption that the market for the product or the transportation facilities are such that the yearly production can be increased to the desired amount.

Specifically, the carrying charge is considered herein as yearly expense, such as state or local taxes, which are assessed each year against the total tonnage remaining in the ground (or the total coal acreage remaining) at the beginning of the year, at a fixed rate per ton (or a fixed rate per acre) or a fixed percentage of the assessed value, in the latter case it being necessary to convert such percentage into a rate per ton or per coal acre. In this connection it may be pointed out that if the tax (or other expense) is assessed on the basis of yearly production (that is, a fixed rate per ton or per acre extracted) or on the basis of yearly gross or net income (such as Federal income tax), such charges for each year take on the status of ordinary operating expenses which are chargeable in their entirety against the production of that year and, therefore, no part of such expense is assignable to any other year's production. In this latter case, there would be no carrying charge, and coal in the ground, if any of it had value, would have some value however small, no matter how long an interval elapsed before it was extracted. On the other hand, a yearly tax or expense that is attached to the coal in the ground before it is extracted and is repeated every year until the coal is mined is an expense that takes on an entirely different status, and it is incorrect to assign to the production of that year as operating expense the total paid in the same year. Of the amount of tax paid, therefore, in each year (if each dollar paid in one year is to be identified as different from each dollar paid in any other year) only a small portion is assignable to the production of the same year and the remainder should be capitalized at a suitable interest rate and assigned to or distributed over the production of later years.

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In order to give effect to this general principle, it has been the practice in the industry, when an analytical appraisal (valuation by discounting expected profits to present worth) is being considered, to assume what is called a normal life of, say, 40 years (in some cases it is understood the so-called normal life is taken as low as 30 years and in some cases as high as 50 years) and hold that any coal that cannot be mined until after 40 years will add no value to the deposit. It is the understanding of the writer that the life taken as the normal life is not the result of computation based on a known formula but is rather an estimate of the average life, based on experience, beyond which coal cannot in general be mined at a profit. The formula given hereinafter, which is very simple, will definitely answer this question as to each specific case, if the relation between the per ton expected profit, and the per ton carrying charge (also the interest rate to be used in capitalizing the carrying charge) is known, the required life being independent of all other factors.

As to the plant capacity most suitable to give the most economic life, the theory carried into effect in this paper is based on the assumption that the cost of the plant is a direct function of the rate of yearly production; that is to say, if the yearly production is doubled, the cost of the plant including development is also doubled. Furthermore, this assumption applies only to preliminary expenditures (or those expenditures which must be made before operations can begin) it being further assumed that the cost of renewals, replacements, construction of new plants or parts of plants and all development work which are required during the operating life are to be treated as operating expense as if they were to be distributed uniformly over the life. While these assumptions may not be in some cases an accurate measure of these relations as shown by actual experience, they are, it is believed, accurate enough for the purpose of the analysis and computations covered herein. If it is shown, for example, that the increase in cost of the preliminary plant, including the necessary preliminary development costs, is not directly proportional to the increase in production, some relation must be assumed between plant costs and yearly production and this relation may be assumed to be different from that found by direct proportion. In this case it may be found that if the production increases *p* per cent the plant costs increase only ¾ of *p* per cent, and this relation may be given effect in the formulas derived hereinafter. As to the assumption that the cost of all renewals, replacements, new plant construction and development will be uniformly distributed over the operating life, it is obvious that such will not be the actual experience, as in some years major plant construction may be necessary, the cost of which will exceed the current costs of other years. When such outlays for future expenditures can be predicted they can be given effect in any valuation formula, but in ordinary cases the difference in the results will not be enough to affect materially the problems covered herein. Accordingly, the operating profits as used herein represent the profits after all future plant and development requirements are deducted but before carrying charges or depreciation are deducted.

In the presence, then, of a decreasing income as the years elapse owing to the accumulation of the carrying charges, the problem presented may be specifically described as follows (where the tonnage reserve is the same):

1. As the life is increased the preliminary plant cost is decreased, which tends toward a higher value for coal exclusive of plant.
2. As the life is increased the present worth factor decreases, which tends toward a reduction in value.
3. As the life is increased, the total net profits (after carrying charges are deducted) decreases, which tends toward a lower value.

In the presence of these opposing influences a life can be determined which will, after the preliminary plant costs are deducted, represent the highest or the maximum value of coal in place (value of deposit exclusive of plant). If a shorter life is taken the value of the coal in place will be less, and if a longer life is taken the value of the coal in place will also be less. It is assumed herein that the life thus determined for the maximum value of coal in place is the most economical life. After the life is determined which will yield the greatest value for the coal as described, the investor may find the expenditure for plant requirements is greater than he feels justified in making as an expenditure preliminary to the beginning of operations, but the determination places before him the necessary knowledge, and his decision as to the amount he is willing to invest in plant and equipment may be made in the light of such knowledge.

With this preliminary statement we may now proceed with the analysis and derivations.

Notation

Let:

T = tonnage reserves; known.

t = yearly production in tons; unknown.

n = life of reserves in years; unknown.

$T \div t = n$ or $t = T \div n$.

P = preliminary plant cost (including development) in dollars per ton per year; known.

$Pt = PT \div n$ = total initial plant cost; unknown.

c = carrying charge per ton per year; known.

cT = total carrying charge paid in first year; known.

$c(T - T \div n)$ = total carrying charge paid in second year; unknown.

$c(T - 2T \div n)$ = total carrying charge paid in third year; unknown.

$c(T - 3T \div n)$ = total carrying charge paid in fourth year; unknown.

$c(T - kT \div n)$ = total carrying charge paid in $(k + 1)$th year; unknown.

$c\{T - (n - 1)T \div n\}$ = total carrying charge paid in nth year; unknown.

r = risk rate of interest; $R = 1 + r$; known.

e = safe rate of interest; $E = 1 + e$; known.

I = total operating profits after future plant and development costs are deducted, but before carrying charges or depreciation are deducted; known.

p = operating profit per ton, whence $pT = I$; known.

$I \div n$ = annual operating profits, assumed uniform throughout life; unknown.

V = present worth value of expected net profits (includes initial plant cost); unknown.

M = value of coal in place, exclusive of initial plant cost (usually referred to as "coal only") or $M = V - PT \div n$; unknown.

In the above notation, it will be noted that provision is made for two rates of interest and they are introduced as a preliminary to the consideration of both types of valuation discount formulas, one being referred to as "single rate" (or straight compound interest, or straight annuity) and the other as "double rate" (Hoskold type) formula. The distribution of the carrying charge, however, is independent of the type of formula, it being necessary only to know the interest rate which, being known, will give the same distribution regardless of the formula type.

DISTRIBUTION OF CARRYING CHARGES

Each year the carrying charge is assumed to be assessed against each unit (ton or coal acre) remaining in the ground at the beginning of the year and presumably paid in that year. These amounts as paid, however, should not be charged in their entirety against the production or profits of that year, but a portion thereof should be capitalized, and draw interest until each year's payment is properly distributed, so that the portion of the tonnage remaining in the ground which is extracted each year shall have assigned to it the accumulated portions of yearly carrying charges paid in previous years plus interest accretions; and the total thus determined for each year taken as the expense for carrying charges for that year. For instance, in the kth year there will be paid as carrying charges $c\{T - (k-1)T \div n\}$. A portion of this should be charged against future years and there should be charged against the kth year the accumulated portions of the carrying charges paid in previous years, plus interest accretions. Thus, in the first year there was paid for the benefit of the tonnage extracted in the kth year one nth of the total carrying charge paid in first year, or $cT \div n$; in the second year there was paid for the benefit of the tonnage extracted in the kth year one $(n-1)$th of the total carrying charge paid in the second year, or $c \div (n-1)(T - T \div n)$; in the third year there was paid for the benefit of the tonnage extracted in the kth year one $(n-2)$th of the total carrying charge paid in the third year, or $c \div (n-2)(T - 2T \div n)$; and so on until the kth year. In the kth year there was paid for the benefit of the tonnage extracted in the kth year one $(n-k+1)$th of the total carrying charge paid in the kth year, or $c \div (n-k+1)\{T - (k-1)T \div n\}$. There was paid, therefore, in any year k and in all previous years for the benefit of the tonnage extracted in the kth year the total as indicated below:

$$\frac{c}{n}T + \frac{c}{n-1}\left(T - \frac{T}{n}\right) + \frac{c}{n-2}\left(T - 2\frac{T}{n}\right) + \cdots$$
$$\frac{c}{n-k+1}\left\{T - (k-1)\frac{T}{n}\right\}$$

or $$\frac{cT}{n} + \frac{cT}{n} + \frac{cT}{n} + \frac{cT}{n} + \cdots \frac{cT}{n},\ k \text{ times or } \frac{kcT}{n}$$

Thus, it may be stated that in any year k (less than n) and in all previous years there was paid in carrying charges for the benefit of the tonnage extracted in the kth year the amount $kcT \div n$ dollars, or that in each year up to and including the kth year there was paid for the benefit of the tonnage extracted in the kth year one nth of the total carrying charge paid in the first year, all of which is fairly obvious from the original statement of the problem.

As to interest on these advance payments, if the rate r is selected, the interest accumulations on the amount $cT \div n$ paid in the first year for the benefit of the tonnage extracted in the kth year will be at the rate r, compounded annually from the end of the first year to the end of the kth year, and this accumulation including the amount paid will be represented by $(cT \div n)R^{k-1}$. Similarly the amount, $cT \div n$, paid in the second year for the benefit of the tonnage extracted in the kth year will, at the end of the kth year with interest accumulations, amount to $(cT \div n)R^{k-2}$; the amount, $cT \div n$, paid in the third year for the benefit of the tonnage extracted in the kth year will, at the end of the kth year with interest accumulations, amount to $(cT \div n)R^{k-3}$ and so on, so that the total amount paid in the kth year and in all previous years, plus interest accretions, all for the benefit of the tonnage extracted in the kth year, will be;

$\frac{cT}{n}R^{k-1} + \frac{cT}{n}R^{k-2} + \frac{cT}{n}R^{k-3} + \cdots \frac{cT}{n}R^0$ which is a geometrical series whose sum will be found to be

$$\frac{cT}{n}\,\frac{R^k - 1}{r} \qquad [1]$$

The above expression, therefore, represents for any year k the carrying charge expense plus interest accretions, which may be regarded as the carrying charge expense assignable to that year. As this expression covers any year, by substituting the year number for k we derive the final distribution by years as shown by the year by year series given below:

1st year: $\frac{cT}{n} \cdot \frac{R - 1}{r} = \frac{cT}{n}$

2d year: $\frac{cT}{n} \cdot \frac{R^2 - 1}{r}$

3d year: $\frac{cT}{n} \cdot \frac{R^3 - 1}{r}$

kth year: $\frac{cT}{n} \cdot \frac{R^k - 1}{r}$

nth year: $\frac{cT}{n} \cdot \frac{R^n - 1}{r}$

An inspection of the above clearly shows that the expense assigned each year, as the carrying charge, increases as the years elapse, and that a time will come when this yearly expense will overtake, and be greater than, the yearly profits (exclusive of carrying charge), unless such profits also increase at the same or a greater rate. The assumption is made herein that such profits do not increase at all but are realized in equal annual amounts. Therefore, when the carrying charges distributed as shown above are equal to the profits (and in later years are greater), the coal remaining in the ground will have no value or less than no value as at the beginning of the first year. According to the notation, the yearly profit before carrying charges are deducted is represented by $I \div n$. By making this profit equal to the expression for carrying charges in the kth year as shown above and solving for k, a formula results which may be used for the purpose sought, or if;

$$\frac{I}{n} = \frac{cT}{n} \cdot \frac{R^k - 1}{r} \text{ then, } R^k = \frac{Ir}{cT} + 1 \text{ and}$$

$$k \log R = \log \left(\frac{Ir + cT}{cT}\right) \text{ and } k = \frac{\log (Ir + cT) - \log cT}{\log R} \qquad [2]$$

Formula 2 may be simplified still further by using $pT = I$. Make this substitution in formula 2 and

$$k = \frac{\log (pTr + cT) - \log cT}{\log R} = \frac{\log T + \log (pr + c) - \log c - \log T}{\log R}$$

or finally

$$k = \frac{\log (pr + c) - \log c}{\log R} \qquad [3]$$

Formula 3 is shown to depend only on profit per ton p, the per ton carrying charge c and the interest rate r, and is independent of all other factors. A study of formula 3 will show that the value of k is dependent only on the relation p bears to c and the interest rate r, for if a relation h is assumed so that $p = hc$, the formula is further simplified and $k = \log (hr + 1) \div \log R$. The use of this latter formula, however, would involve the necessity for determining the relation between p and c by dividing p by c and no particular advantage is gained, unless the relation can be readily determined either by inspection or by one or two or possibly three figures.

As before indicated, formula 3 is also independent of the type of valuation formula. If the double rate formula is used (Hoskold type) and e is the interest rate selected for capitalizing the carrying charges, then substitute e for r in formula 3 and determine k. If the risk rate r is used, being the same in each of the valuation formula, k will be the same regardless of the valuation formula used.

Numerical Example

Assume the per ton profit before deducting carrying charge to be \$0.25, the per ton carrying charge as \$0.002 and the interest rate 8 per cent. Substitute in formula 3 and

$$k = \frac{\log (0.25 \times 0.08 + 0.002) - \log 0.002}{\log 1.08} = 31 \text{ years.}$$

In this numerical example it is shown that after 31 years the unmined coal has less than no value as at the beginning of the first year. If for economic reasons or by reason of limited transportation facilities, or for any other reason, there can be no increase in production above a certain yearly tonnage, which is too low to extract the tonnage reserve owned or held within the 31 years determined, the excess is a liability rather than an asset, and this should be enough to convince the taxing authority that the tax should be eliminated, otherwise a disposal of such excess, or allowing it to be sold for unpaid taxes, or turning it back to the public domain, as the case may be, would be justifiable.

On the other hand, if production can be increased so that the tonnage may be extracted within the 31 years, the entire tonnage may be retained, and the question then arises as to the amount of the increase in production and in the plant cost which follows.

In either case the point is now reached where the question of the specific value is to be considered; in the former case the value of the coal that can be extracted in 31 years, eliminating the remainder as valueless, and in the latter case the value of all of the coal. The valuation formula based on one rate of interest is considered first.

VALUATION, ONE INTEREST RATE

For convenience in presentation, the valuation or discount formula where one interest rate only is involved (straight annuity law based on compound interest), is derived by regarding the same on a sinking fund basis; and where only one rate is used, the same formula results regardless of whether or not the same is considered from the standpoint of a sinking fund, from the standpoint of diminishing capital, or from the standpoint of the sum of the several present worth values of each year's income considered separately. Furthermore, the double rate formula to be considered later is essentially a sinking fund formula and for comparison purposes it is better that both be made subject to the same analysis.

The sinking fund theory, as applied to valuation or discount formulas, declares that the investor is entitled to interest on his total investment each and every year during the investment period and that the excess of the yearly income over such yearly interest shall be placed in a sinking fund drawing interest at the same rate, compounded, from the date contributed to the sinking fund until the end of the investment period, so that at the end of such investment period there shall be accumulated in sinking fund contributions and interest accumulations an amount exactly equal to the original investment. Giving effect by years to this theory and to the decreasing income caused by the increasing carrying charges as the years elapse (as shown hereinbefore), the following year by year series shows the sinking fund accumulations:

$$\begin{array}{ll}
\text{1st year} & \left\{\frac{I}{n} - Vr - \frac{cT}{n}\cdot\frac{R-1}{r}\right\}R^{n-1} \\
\text{2nd year} & \left\{\frac{I}{n} - Vr - \frac{cT}{n}\cdot\frac{R^2-1}{r}\right\}R^{n-2} \\
\text{3rd year} & \left\{\frac{I}{n} - Vr - \frac{cT}{n}\cdot\frac{R^3-1}{r}\right\}R^{n-3} \\
k\text{th year} & \left\{\frac{I}{n} - Vr - \frac{cT}{n}\cdot\frac{R^k-1}{r}\right\}R^{n-k} \\
n\text{th year} & \left\{\frac{I}{n} - Vr - \frac{cT}{n}\cdot\frac{R^n-1}{r}\right\}R^{0}
\end{array}$$

Under the definition given, the sum of the terms of above series must be equal to V, or

$$V = \left\{\frac{I}{n} - Vr\right\}(R^{n-1} + R^{n-2} + R^{n-3} + \cdots 1) + \frac{cT}{nr}(R^{n-1} + R^{n-2} + R^{n-3} + \cdots 1)$$

$$-\frac{cT}{nr}\left\{R^n + R^n + R^n + \cdots R^n, n \text{ times}\right\} \text{ or}$$

summing the bracket series and

$$V = \left(\frac{I}{n} - Vr\right)\frac{R^n - 1}{r} + \frac{cT}{nr}\cdot\frac{R^n-1}{r} - \frac{cT}{nr}nR^n.$$

Solve for V and obtain:

$$V = \frac{I}{n}\cdot\frac{R^n-1}{rR^n} + \frac{cT}{nr}\cdot\frac{R^n-1}{rR^n} - \frac{cT}{r} \text{ or}$$

$$V = \frac{R^n-1}{nrR^n}\left\{I + \frac{cT}{r}\right\} - \frac{cT}{r} \qquad [4]$$

Formula 4 represents the present worth value of the expected profits, which includes of course the original plant investment. The value exclusive of the initial plant investment would be $M = V - PT \div n$ where M represents the value of coal only (see Notation) or:

$$M = \frac{R^n-1}{nrR^n}\left\{I + \frac{cT}{r}\right\} - \frac{cT}{r} - \frac{PT}{n} \qquad [5]$$

Numerical Examples

Let $T = 8{,}000{,}000$ tons, $I = \$2{,}000{,}000$ or $p = I \div T = \$0.25$, $P = \$1.00$, $c = \$0.002$, $r = \$0.08$, $R = 1.08$.

First let $n = 31$ years, or the life during which the 8,000,000 tons must be extracted in order that the yearly carrying charge, distributed as shown, shall not exceed the profits. Make the numerical substitutions indicated in formula 5 and

$$M = \frac{1.08^{31} - 1}{31 \times 0.08 \times 1.08^{31}}\left\{2{,}000{,}000 + \frac{0.002 \times 8{,}000{,}000}{0.08}\right\} - \frac{0.002 \times 8{,}000{,}000}{0.08} - \frac{8{,}000{,}000}{31}$$

whereupon $M = \$347{,}405.09$, with a plant investment of $8{,}000{,}000 \div 31 = \$258{,}064.52$, or $V = \$605{,}469.61$.

If the tonnage of 8,000,000 cannot be extracted within the 31 years, or at the rate of 258,064.52 tons yearly, let it be assumed that the maximum production is 200,000 tons yearly, or that it will require 40 years to extract 8,000,000 tons. Formula 3 tells us that the tonnage unmined after 31 years will have less than no value. At 200,000 tons a year for 31 years the tonnage that could be mined at a profit would be 6,200,000 tons, the remaining 1,800,000 tons having less than no value. Substitute in formula 4 $T = 6{,}200{,}000$ and $I = \$1{,}550{,}000$ in lieu of 8,000,000 tons and \$2,000,000 respectively:

$$V = \frac{1.08^{31} - 1}{31 \times 0.08 \times 1.08^{31}}\left\{1{,}550{,}000 + \frac{0.002 \times 6{,}200{,}000}{0.08}\right\} - \frac{0.002 \times 6{,}200{,}000}{0.08}$$

Whereupon $V = \$469{,}238.97$ including the initial plant cost; and the initial plant requirement cost would be $6{,}200{,}000 \div 31 \times \$1.00 = \$200{,}000$.

If 32 years is taken for n and 200,000 tons yearly is still assumed as the maximum production the total tonnage would be 6,400,000 and the total profits would be \$1,600,000; and even though a greater total operating profit I is taken (\$1,600,000 as compared with \$1,550,000) our analysis has told us that a value less than \$469,238.97 would result.

Make these latter numerical substitutions and

$$V = \frac{1.08^{32} - 1}{32 \times 0.08 \times 1.08^{32}}\left\{1{,}600{,}000 + \frac{0.002 \times 6{,}400{,}000}{0.08}\right\} - \frac{0.002 \times 6{,}400{,}000}{0.08}$$

Whereupon $V = \$468{,}924.98$, or less than that for 31 years as claimed. The initial plant requirement would of course be the same, or \$200,000.

We now proceed to the consideration of an increase in plant capacity and rate of yearly production, with a view to determining the maximum value of the coal after the plant costs are deducted.

Formula 5 gives the value of M expressed in terms of n and various known factors. It is required then to find such a value for n as will render M a maximum. If the right-hand member of formula 5 is differentiated with respect to n and the first differential coefficient set equal to zero, an expression results which cannot be solved algebraically for n. The determination, then, will require repeated trials ("cut and try" method) for different values for n substituted either in formula 5 or in the differential expression, whichever is the simpler, until the required value is found. In this case the differential expression is the simpler and is shown below:

$$\frac{dM}{dn} = \left\{\frac{I}{r} + \frac{cT}{r^2}\right\}\frac{1}{n^2R^n}\left\{1 + n \log_e R\right\} - \left\{\frac{I}{r} + \frac{cT}{r^2} - PT\right\}\frac{1}{n^2}$$

Make the right-hand member equal to zero; reduce, rearrange, write D for $(I \div r + cT \div r^2)$, and;

$$\frac{D - PT}{D}R^n = 1 + n \log_e R, \text{ or since } \log_e R = \frac{\log_{10} R}{0.43426}$$

$$\frac{D - PT}{D}R^n = 1 + n\frac{\log_{10} R}{0.43426} \qquad [6]$$

As n appears both as an exponent and a coefficient, equation 6 is not algebraically solvable for n and the value of n for numerical values of D, PT and R must be sought by "cut and try" methods.

Numerical Example

Let T = 8,000,000 tons; I = \$2,000,000; P = \$1.00; c = \$0.002 and $r = 0.08$ as before. Make these substitutions in formula 6 and $D = (2{,}000{,}000 \div 0.08 + 0.002 \times 8{,}000{,}000 \div 0.08^2) = 27{,}500{,}000$.

Try 13 years for n and:

$$\frac{(27{,}500{,}000 - 8{,}000{,}000)(1.08^{13})}{27{,}500{,}000} = 1 + 13 \times 0.07696$$

or $1.93 = 2.00$.

Try 14 years for n and

$$\frac{(27{,}500{,}000 - 8{,}000{,}000)(1.08^{14})}{27{,}500{,}000} = 1 + 14 \times 0.07696$$

or $2.083 = 2.077$.

Try 15 years for n and

$$\frac{(27{,}500{,}000 - 8{,}000{,}000)(1.08^{15})}{27{,}500{,}000} = 1 + 15 \times 0.07696$$

or $2.25 = 2.15$; showing that 14 years is the value for n that renders M a maximum.

The same result is obtained by substituting the same numerical values in formula 5, whereupon:

For $n = 13$, $M = \$522{,}177.32$.
For $n = 14$, $M = 524{,}093.35$.
For $n = 15$, $M = 522{,}057.07$.

For a maximum life of 14 years, the initial investment required for plant will be $PT \div n = 8{,}000{,}000 \div 14 = \$571{,}428.57$. The investor has now reached the point where he has before him the necessary information based on which he may decide whether he is willing to make the large investment in plant as indicated in order to realize the maximum benefit financially, or whether he would prefer to reduce the initial investment in plant and lose correspondingly of the financial advantage referred to.

Valuation, Two Interest Rate (Hoskold Type)

The type of valuation formulas derived under this theory is based essentially on the sinking fund theory, and an attempt to regard it from the standpoint of a diminishing capital basis will only lead to violations of the mathematical premises involved. If it is to be regarded from the standpoint of the present worth value of each year's income taken separately, it must still be on a sinking fund basis where the declaration is made that the investor is to receive each and every year interest at the risk rate on his total investment (a constant amount) and that no portion of the capital investment is returnable until the end of the investment period. The theory is precisely the same as that described under the one-rate valuation analysis except that, in the double rate analysis, the sinking fund accumulates interest at a lower rate (safe rate) than the risk rate.

As to the distribution of the carrying charges and interest accretions, the question arises here as to which rate of interest should be taken for the purpose of capitalizing such charges. In support of the higher or risk rate of interest, it may be contended that, since the investor must advance each year the amount to be capitalized, he should receive the same interest return (the risk rate) as that determined for his original investment in mineral and plant, under the assumption that the same risk is involved in making these advances as that found in advancing the money involved in the original investment. On the other hand, it may be contended that, since in general these payments are expected to be made each year from the profits as they are realized, the same risk is not involved as in that of the investment itself, which latter involves the cash payment for the property before any profits are realized. It is the opinion of the writer that the latter view should prevail and that the advanced carrying charges should be capitalized at the safe rate.

However, both views are covered herein and a formula derived for each. In passing, reference may be made to a third contention to the effect that neither the risk rate of interest nor the safe rate fits the conditions and that an intermediate rate should be considered, but it is not believed necessary to cover this refinement.

SAFE RATE FOR CARRYING CHARGES

Giving effect by years to the theory described, using first the safe rate of interest for capitalizing the carrying charges, the following year by year series shows the sinking fund accumulations:

$$\text{1st year}\left(\frac{I}{n} - Vr - \frac{cT}{n}\cdot\frac{E-1}{e}\right)E^{n-1}$$

$$\text{2nd year}\left(\frac{I}{n} - Vr - \frac{cT}{n}\cdot\frac{E^2-1}{e}\right)E^{n-2}$$

$$\text{3rd year}\left(\frac{I}{n} - Vr - \frac{cT}{n}\cdot\frac{E^3-1}{e}\right)E^{n-3}$$

$$k\text{th year}\left(\frac{I}{n} - Vr - \frac{cT}{n}\cdot\frac{E^k-1}{e}\right)E^{n-k}$$

$$n\text{th year}\left(\frac{I}{n} - Vr - \frac{cT}{n}\cdot\frac{E^n-1}{e}\right)E^{0}$$

As before, the sum of the above sinking fund accumulations must be equal to V. Make the required summation and solve for V, whereupon:

$$V = \frac{1}{rE^n - r + e}\left\{\left(I + \frac{cT}{e}\right)\frac{E^n - 1}{n} - cTE^n\right\} \qquad [7]$$

$$M = \frac{1}{rE^n - r + e}\left\{\left(I + \frac{cT}{e}\right)\frac{E^n - 1}{n} - cTE^n\right\} - \frac{PT}{n} \qquad [8]$$

As to the year when the carrying charges overtake the profits, obviously formula 3 can be made to apply by substituting e for r, or

$$k = \frac{\log\,(pe + c) - \log c}{\log E} \qquad [9]$$

Numerical Examples

As to the value of k, assume the per ton profit, $p = \$0.25$, the per ton carrying charge $c = \$0.002$ as before and the interest rate e at 0.04. Substitute in formula 9:

$$k = \frac{\log\,(0.25 \times 0.04 + 0.002) - \log 0.002}{\log 1.04} = 45.7 \text{ years}$$

As to the value V (following the same analysis as under the single rate), assume $T = 8{,}000{,}000$ tons; $I = \$2{,}000{,}000$; $P = \$1$; $c = \$0.002$; $r = 0.08$; as before and $e = 0.04$; $E = 1.04$, and for n first take 45 years, substitute in formulas 7 and 8 and $V = \$385{,}535.97$, $M = \$207{,}758.19$ with an initial plant cost of $8{,}000{,}000 \div 45 = \$177{,}777.78$.

Here the rate of yearly production is 177,777.78 tons. If the maximum yearly production possible is, say, 150,000 tons, then in 45 years there would be extracted only 6,750,000 tons and the total profits I would be \$1,687,500; the remaining tonnage having less than no value. Making these substitutes in formulas 7 and 8 in lieu of 8,000,000 and 2,000,000 with $n = 45$ and all other factors the same, then $V = 325,295.98$, $M = 175,295.98$; with an initial plant cost of \$150,000.

If n is greater than 45 and the maximum yearly production is still held at 150,000 tons (other factors the same), then it is asserted that the value giving effect to the additional tonnage reserve will be less than that for 45 years. Assume 46 years as the life, then the total tonnage reserve will be 6,900,000 and the total profits I will be \$1,725,000. Making these substitutions in formulas 7 and 8, then $V = 324,078.63$ (smaller than for 45 years as claimed), $M = 174,078.63$ with the plant cost of \$150,000.00.

As to an increase in the plant capacity and rate of production such that a maximum value of "coal only" will result, an inspection of formula 8 indicates that differentiation with respect to n and equating to zero will not result in an expression that is algebraically solvable for n, and will not result in an expression that will be in a simpler form than formula 8 itself. Therefore, the "cut and try" may be applied directly to formula 8.

Numerical Example

Let $T = 8,000,000$ tons; $I = \$2,000,000$; $P = \$1$; $c = \$0.002$; $r = 0.08$; $e = 0.04$ and $E = 1.04$. Make these substitutions in formula 8 and .

First try 14 years for n, then $M = \$420,343.91$
Next try 15 years for n and $M = \$421,126.50$
Next try 16 years for n and $M = \$419,345.11$

all of which shows that 15 years is the value of n sought, or that which renders M a maximum. For a life of 15 years, the initial investment required for plant will be $PT \div n = 8,000,000 \div 15 = \$533,333.33$.

Risk Rate for Carrying Charges

Giving effect by years to the double rate theory as described, using the risk rate of interest for capitalizing the carrying charges, the following year by year series shows the sinking fund accumulations:

$$\text{1st year}\ \left(\frac{I}{n} - Vr - \frac{cT}{n}\cdot\frac{R-1}{r}\right)E^{n-1}$$

$$\text{2nd year}\ \left(\frac{I}{n} - Vr - \frac{cT}{n}\cdot\frac{R^2-1}{r}\right)E^{n-2}$$

$$\text{3rd year}\ \left(\frac{I}{n} - Vr - \frac{cT}{n}\cdot\frac{R^3-1}{r}\right)E^{n-3}$$

$$k\text{th year}\left(\frac{I}{n} - Vr - \frac{cT}{n}\cdot\frac{R^k - 1}{r}\right)E^{n-k}$$

$$n\text{th year}\left(\frac{I}{n} - Vr - \frac{cT}{n}\cdot\frac{R^n - 1}{r}\right)E^{0}$$

Summing this series, reducing and solving for V there is obtained:

$$V = \frac{1}{n(rE^n - r + e)}\left\{\left(I + \frac{cT}{r}\right)(E^n - 1) - \frac{cT}{r}\cdot\frac{eR}{(r-e)}(R^n - E^n)\right\} \quad [10]$$

$$M = \frac{1}{n(rE^n - r + e)}\left\{\left(I + \frac{cT}{r}\right)(E^n - 1) - \frac{cT}{r}\cdot\frac{eR}{(r-e)}(R^n - E^n)\right\} - \frac{PT}{n} \quad [11]$$

As to the year when the carrying charges overtake the profits, obviously formula 3 applies, and if the risk rate r is taken at 8 per cent (other factors being the same) the same life will result as that shown by the numerical example following formula 3, or 31 years.

Numerical Examples

The numerical value of k being covered by the preceding paragraph, the value V is next in order. Assume T = 8,000,000 tons; I = \$2,000,000; P = \$1; c = \$0.002; r = 0.08; e = 0.04 and E = 1.04; and for n, first take 31 years, substitute in formulas 10 and 11 and V = 505,527.90, M = 247,463.38, with an initial plant cost of 8,000,000 ÷ 31 = \$258,064.52.

If the maximum yearly production is 200,000 tons, in 31 years there would be extracted only \$6,200,000 tons and the total profits would be \$1,550,000, as in the case of the numerical example under the single rate caption. Making these substitutions in formulas 10 and 11 in lieu of 8,000,000 and 2,000,000, with n = 31, and all other factors the same, V = \$391,784.12, M = 191,784.12, with an initial plant cost of \$200,000.

If n is taken greater than 31 and the maximum yearly production is still held at 200,000 tons (other factors the same), the value covering the additional tonnage will be less than that for 31 years. Assume 32 years as the life, then the total tonnage reserve will be 6,400,000 and the profits I will be \$1,600,000. Make these substitutions in formulas 10 and 11 and V = 388,568.01 (smaller than for 31 years claimed), M = 188,568.01, with the initial plant cost of \$200,000 as before.

As to an increase in plant capacity and production such that a maximum value of "coal only" will result, an inspection of formula 11 indicates that differentiation with respect to n and equating to zero will not yield an expression that is algebraically solvable and will not yield an expression that will be in a simpler form than formula 11 itself. Therefore, the "cut and try" method may be applied directly to formula 11.

Numerical Example

Let $T = 8{,}000{,}000$ tons; $I = \$2{,}000{,}000$; $P = \$1$; $c = \$0.002$; $r = 0.08$; $e = 0.04$ and $E = 1.04$. Make these substitutions in formula 11 and

First try 13 years for n, then $M = \$404{,}177.06$
Next try 14 years for n, then $M = \$406{,}692.78$
Next try 15 years for n, then $M = \$405{,}658.87$

all of which shows that 14 years is the value of n that renders M a maximum. For a life of 14 years the initial investment required for plant will be $PT \div n = 8{,}000{,}000 \div 14 = \$571{,}428.57$.

In considering the foregoing, it is recognized that many conditions which actually exist may be cited, particularly those of an economic character, to which the principles and theories covered herein are clearly inapplicable, and in some cases which, when an attempt is made to apply such principles, will result in an apparent contradiction of the known effects. For instance, it may be contended that, often it is the case, a large operator having an established market for a certain yearly production may own or control large tonnage reserves which the test covered by formula 3 indicates will exceed that which can be mined at a profit but which, nevertheless, the operator must retain in order to eliminate a competitor operator in the same field, thus indicating that the coal that the test shows cannot be mined at a profit must have a value, and thus apparently contradicting the theory. A moment's consideration of such a condition will disclose that, rather than a contradiction, the theory is not only directly applicable but is essential in order that the investor may have at his command the knowledge necessary, in the presence of such conditions, to arrive at a proper valuation. For, if it is necessary to retain coal lands which cannot be mined at a profit in order to eliminate competition, it is necessary to retain such lands in order to realize the profits I on the coal which can be extracted prior to the time when the carrying charge overtakes such profits, and if the excess coal were disposed of and a competitor established, presumably the profits estimated could not be realized in the presence of such competition. Thus a condition is attached to the estimate of profits, which is that certain lands must be retained which have less than no value in order that other lands may yield a profit. It would, therefore, be incorrect to base the present worth value on these same yearly profits, $I \div n$ which are realized only during the time when such profits are greater or equal to the carrying charges (and excluding the excess tonnage), for such a value would be correct only in the event that the estimate of such profits is free from any condition which at the valuation date is known to affect the probability of their realization. If it is clearly established that certain excess coal lands containing a certain excess tonnage must be retained in order to realize a certain per ton profit (before carrying charges are deducted), then the present worth value must be based on the entire tonnage (including the excess tonnage) and effect must be given to the negative profits and negative value of such excess. The value would, therefore, be lower than that in the absence of such conditions by an amount depending upon the excess tonnage and the number of years required for extraction.

For a numerical example, take the present worth value V of $469,238.97 determined under the single rate theory based on a yearly production of 200,000 tons at 25¢ for 31 years, or on a total tonnage of 6,200,000 and a total profit of $1,550,000. Here the present worth value of $469,238.97 is predicated on the assumption that no condition is attached to the estimate of 25¢ per ton and that the excess tonnage (8,000,000 − 6,200,000) of 1,800,000 would not be retained. If it must be retained in order to realize the 25¢ per ton profit, the present worth value would be based on the same yearly profits extended over a life of 8,000,000 divided by 200,000, or 40 years, using the same formula 4, or:

$$V = \frac{1.08^{40} - 1}{40 \times 0.08 \times 1.08^{40}}\left(2,000,000 + \frac{0.002 \times 8,000,000}{0.08}\right) - \frac{0.002 \times 8,000,000}{0.08}$$

or V = $455,853.75 with an initial plant cost of $200,000.

In conclusion, it may be stated that the principles set forth herein, while peculiarly applicable to coal valuations, are also applicable to copper, iron or any other mineral or metal or timber, if subject to the same character of taxation.

DISCUSSION

(*Howard N. Eavenson presiding*)

J. B. Dilworth,* Philadelphia, Pa. (written discussion).—Mr. Craigue's paper is a striking example of the application of higher mathematical principles to practical valuation problems. While the mathematics involved are probably too difficult for the average engineer to follow in detail, it is not necessary that he should do so in order to profit by the results obtained. The general principles involved have been employed by valuation authorities in the past and widely accepted as applicable to such appraisals as are here considered.

The validity of the method of determining present values of mining properties by discounting prospective profits has been recognized almost without exception by the members of the Institute's committee on Methods of Valuing Coal Properties. In fact, no other scientific method seems possible. The main difficulty in its application lies in the proper determination of certain essential factors, such as average future profits per ton and annual production, used in the calculations; hence the necessity for checking results of such calculations by contemporary regional prices of similar properties.

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The method here set forth of determining the effect of plant capacity and carrying charges on coal-land values is important to at least two classes of interests: First, to the prospective operator of a coal property in enabling him to decide on the plant capacity that will give the largest return on his capital investment, or, if his output is limited by other considerations, in indicating the amount of tributary coal his plant should have available to afford the greatest profit; second, to the taxing authorities in enabling them properly to assess areas of coal land either wholly or partly subject to mining development. Were these principles more generally recognized, serious injustice to coal-land owners in appraisals for taxation purposes would be less frequent than at present.

As this paper is an exposition of general principles and the numerical examples given are merely illustrative of the application of these principles, it is hardly permissible to consider certain figures used in the examples as indicative of the author's opinion of what values should be used in actual cases. However, it is at least interesting to note that he is in accord with the opinions believed to be generally held by appraisers; namely, that two interest rates, a "safe" and a "risk" rate (Hoskold type formula), should be used in valuations based on prospective earnings rather than the single rate or annuity formula; and also that in illustrating the double rate method he uses 4 per cent for the "safe" rate and 8 per cent for the "risk" rate, which is also in accord with the views of numerous engineers.

Again, the fact is brought out in the numerical examples that 14 or 15 years is the operating life that gives the maximum value to the tributary coal; also that in periods varying from 31 to 45 years, according to interest rates used, the carrying charge per ton of coal will equal the profit per ton and hence the value of the coal yet to be recovered by the mine plant is zero or less. These results are obtained with factors of profit, plant cost and interest rates that would have been applicable to many actual coal operations in times past. While it is not quite true, as assumed here, that preliminary or initial plant cost varies directly with capacity, the deviation from this assumption in actual practice is hardly enough to invalidate the results obtained from the formulas.

The desirability of short mine life from the standpoint of economy of operation has been brought out in earlier Institute papers and elsewhere. Yet it seems rarely to be appreciated by the practical operator, and few coal plants of considerable size have been designed for lives as short as even 20 years.

Mr. Craigue is to be congratulated on his paper and the Institute is indebted to him for his masterly presentation of a phase of the valuation subject none too well appreciated or understood by mining engineers and operators generally.

Competitive Relation of Coal and Petroleum in the United States*

By W. Spencer Hutchinson,† Cambridge, Mass., and August J. Breitenstein,‡ Ashland, Pa.

(New York Meeting, February, 1933)

The outstanding engineering accomplishment of the last three decades has been the development and application of more and cheaper power and its use instead of the labor of men and animals. Substitution of power for manual labor has taken effect on every hand in the United States. With the aid of energy mechanically available to his hand, one man does the work that only a few years ago required two, three or even a greater number of laborers.

The results in respect to the favorable economic effects are well known; namely, increased production, higher wages and lower costs. On the other hand, the vast social effects, both good and bad, have been none the less important, even if less understood and only casually regarded.

The chief sources of energy today are the mineral fuels, coal and petroleum. In the United States they yield more than 90 per cent of the total energy demand. Thirty years ago the proportion of energy supplied by the mineral fuels was likewise above 90 per cent, but the division according to origin was very different. These relations are shown for the years 1900, 1930 and 1950 in Table 1.

Method of Preparation and Interpretation of Statistical Data

The fuels comprised in this study are: (1) the coal group including both bituminous and anthracite coal; (2) the petroleum group, taking in not only petroleum but natural gas and natural-gas gasoline. Natural gas for many years has been used in localities relatively near the wells but statistics were found no earlier than 1910. Demand has rapidly

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* The information and statistical data upon which this paper is based were derived in the main from a thesis research by August J. Breitenstein entitled "Mineral Fuel Supplies and Markets in the United States," done under the direction of W. Spencer Hutchinson and submitted in partial fulfillment of the requirement for the degree of Master of Science at the Massachusetts Institute of Technology. The thesis was offered by Mr. Breitenstein for publication but was found too expensive to print because of its 31 extensive tables, 7 charts and 150 pages of typed text and bibliography.

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TABLE 1.—*Derivation of Energy in the United States*

Sources	Allocation of Total Energy Derived, Per Cent		
	1900	1930	Forecast 1950
Coal—bituminous and anthracite	91.3	60.3	46.6
Petroleum and its products, including also natural gas and natural-gas gasoline	4.3	31.6	45.3
Hydro-power	4.4	8.1	8.1
Total	100.0	100.0	100.0

increased as a vast network of pipe lines has been created so that the gas can be delivered to the industrial cities instead of wasted into the air at the wells. The production of natural-gas gasoline first appears in the statistics for 1910.

Hydro-power contributes its appropriate share of the total energy demand and accordingly is added to the quantity supplied by the fuels, and so becomes a factor in the statement and solution of the problem.

In order to compare and interpret the significance of the consumption of the different fuels, a common unit of measurement is required. Not all can be expressed in tons or barrels or volume, and recourse was accordingly had to the energy derivable, which has been determined according to efficiency of use of the respective fuels, both with relation to time and according to character of use. No record has been found of any previous attempt to forecast energy demand of the United States based on "effective" energy utilized and derived from the fuel consumed, and other sources.

Inasmuch as energy demand is independent of any one of the fuels, it is likewise independent of the supplies or cost of any single fuel just so long as there is abundant and widely distributed supply of one or more of them. This principle permits a study of total energy demand based on statistical data relating to the sources of supply in the years already past; and from these data forecast of future demand may be made, for the energy graph shows a consistent relation to the statistics of population. The curve of total energy demand (Fig. 1) fulfills the aforesaid condition, and accordingly is given the name of "master curve."

The meaning of this is made plain by pointing out that the sum, in any year, of energy demand from coal, petroleum, natural gas, natural-gas gasoline and hydro-power invariably equals the total energy demand for that year. This statement omits consideration of cord wood, peat, windmill and animal power, because of lack of statistics, but the omissions are believed to affect the results only to a small degree and are known not to invalidate the deductions and forecast. Energy derived from petroleum coke has been included with petroleum.

Considerable literature was found on the subject of thermal efficiencies relating to the period under review, and was interpreted by Mr. Breitenstein. Thermal energy is measured by deducting the energy wasted from the total energy contained in the fuel as expressed in British thermal units.

The term "thermal efficiency" is used to express the percentage factor by which total energy has been converted to effective energy. Thermal efficiency of all the fuels in 1930 was found to vary from 5 to 80 per cent, according to the different fuels and the different purposes and methods of use. Computed weighted averages for the same year vary from 20 to 45 per cent, according to the different fuels; the minimum weighted average of thermal efficiency was estimated to be 12 per cent in the year 1840.

The determination of thermal efficiency and its effect on the trend of energy demand proved to be one of the most obscure problems in this study, and even though space here will not permit of going further into the subject, it is believed the estimates of trend have been sufficiently established to impart validity to the subsequent forecasts.

Statistics of production of the different classes of mineral fuel were reduced to quantities consumed by making allowance for exports and imports and for stocks at the beginning and end of each year. Exports were not included because they did not, of course, provide energy in the United States. The quantity of exports nevertheless adds to the market for fuels produced in the United States, and the study of the export market and the determination of its trend is another story which is not attempted here.

TABLE 2.—*Consumption of Mineral Fuels in the United States*

Fuels	Years					
	1850	1870	1890	1910	1930	Forecast 1950
Bituminous coal[a]	2,880	17,719	110,785	407,257	449,094	444,000
Anthracite coal[a]	4,138	15,528	45,597	81,110	67,925	55,500
Petroleum[b]	None	2,553	30,874	160,000	868,484	1,419,000
Natural gas[c]	None	None	None	509,155	1,940,000	3,800,000
Natural-gas gasoline[b]	None	None	None	100	51,762	126.000
Hydro-power[d]	622	1,205	1,555	4,650	13,808	

[a] Thousands of tons of 2000 lb.
[b] Thousands of barrels.
[c] Millions of cubic feet.
[d] Thousands of horsepower plant capacity.

The results of the statistical studies and the tabulation are shown in Table 2, summarized and reduced to the simplest terms. Figures of mineral fuels consumed in the United States are all actual, each being derived from statistical records of the year; the only exception is for

1950, which is a forecast derived from data developed in Table 3, by reversing the process of computation, and making provision for forecast improvements of practices and efficiencies in the use of the respective fuels.

TABLE 3.—*Sources of Energy in the United States*

	Years					
	1850	1870	1890	1910	1930	Forecast 1950
EFFECTIVE ENERGY SUPPLY IN TRILLIONS OF BRITISH THERMAL UNITS						
Coal	32	173	955	3,620	5,443	6,760
Petroleum	None	3	41	405	2,857	6,570
Hydro-power	15	32	53	316	731	1,170
Total	47	208	1,049	4,341	9,031	14,500
POPULATION AND ENERGY						
Population (millions)	23.2	38.6	62.9	92.0	122.8	154.
Energy per capita (millions of B.t.u.)	2.	5.	16.	47.	73.	94.

The figures in Table 3 show the totals for the different mineral fuels expressed in terms of effective energy. Statistical data relating to horsepower of developed water power have been converted to effective B.t.u. equivalent by means of computing corresponding quantity of coal required in agreement with the conditions for the year under review and multiplying by the appropriate efficiency factor. The two lower lines show the population of the United States at the dates chosen, and the per capita demand for energy expressed in millions of British thermal units.

Fig. 1 was drawn from the data shown in Table 3 as far as 1930. The forecasts are derived by computations according to appropriate exponential rates of annual increase, and the results so determined were plotted as shown by projection of the lines of population and total energy demand. Both of these quantities show consistent growth for the past 30 years without change of trend. Energy from coal, energy from petroleum and energy from hydro-power must, in the nature of the premises upon which the chart is constructed, be equal to the ordinate of the master curve which expresses total energy demand. The validity of the forecasts is revealed graphically by the chart.

In respect now to Table 2, the quantities of coal and petroleum forecast are determined according to the effective energy factors appropriate to the circumstances, and the figures in the right-hand column are expressive of the demand for these fuels in 1950. It will be observed that forecast for hydro-power has not been made. The equivalents in British thermal units instead of water power have served their purposes in preparing the chart and making the other forecasts, and no attempt is made here to reverse the process and predict hydropower supply.

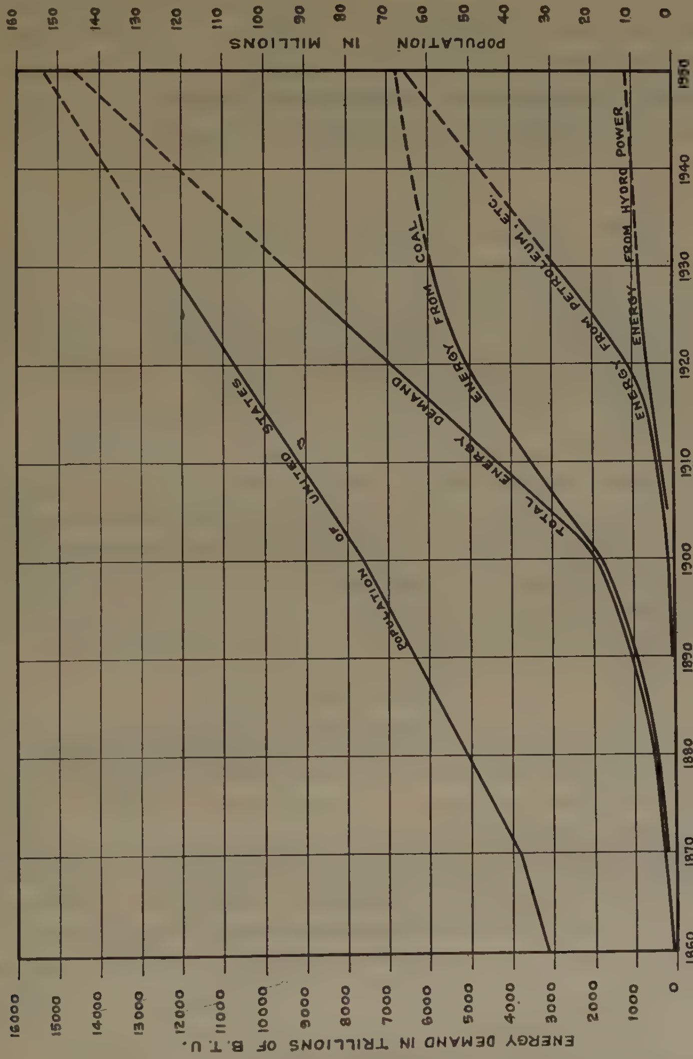

FIG. 1.—ENERGY MARKET COMPETITIVE RELATION AND FORECAST FOR COAL AND PETROLEUM IN THE UNITED STATES.

FUTURE OF ENERGY DEMAND PER CAPITA OF POPULATION

Table 3, in the last line, gives the total energy demand per capita and shows a consistent and steady growth, but when the demand for coal and the other mineral fuels is examined on a per capita basis, we find a very different story.

The figures in Table 4 show a progressive and increasing rate in the consumption of bituminous and anthracite coal from 1850 to 1910.

Actually, if the years intervening between 1910 and 1930 are examined, the per capita consumption of coal ran up to 6.08 tons in 1917, which was the maximum. From then on, the decline in per capita consumption of coal was rapid. It reached 4.88 tons in 1929, and 4.21 tons in 1930. The forecast for 1950 is 4.16 tons; this might seem inconsistent unless it be remembered that the year 1930 was a year of depression when the industries took less than the normal quantity of coal.

TABLE 4.—*Consumption of Fuels*

Fuels	Years					
	1850	1870	1890	1910	1930	Forecast 1950
Coal, tons of 2000 lb. per capita	0.30	0.86	2.49	5.31	4.21	4.16
Petroleum, bbl. per capita	None	0.07	0.49	1.74	7.07	12.47
Natural gas, M. cu. ft. per capita	None	None	None	5.53	15.80	31.17
Natural-gas gasoline, bbl. per capita	None	None	None	0.001	0.42	1.06

Table 4 also shows the rapid rise of petroleum and related mineral fuels as sources of energy. These fuels have not only discovered new markets, particularly in the field of automotive transportation, but have made, and are making, incursions into fields hitherto for many years occupied exclusively by coal. The high point of production of petroleum lies in the future.

IN CONCLUSION

The demand for energy, irrespective of its source, increases steadily at an exponential rate considerably greater than the rate of growth of the population as, for example, in the decade 1920 to 1930; the exponential rate of growth of population was 1.5 per cent annually, whereas the rate found in respect to energy demand was 3.5 per cent between 1920 and 1925; 3.0 per cent between 1925 and 1930. Such analysis of the growth factors indicate that at some time, a good many years in the future, the demand of the people for energy will become satisfied in the industrial areas, and from then on its increase may be only in proportion to the increase in population.

If, on the other hand, comparison is made of the exponential rates of energy derived respectively from coal and petroleum, the former is growing at a much slower rate than the population; whereas petroleum during the five-year period 1925 to 1930 progressed at an annual exponential rate of 7.0 per cent.

If we accept the estimates of petroleum reserves expected to be available at low cost for 20 years, the outlook for the coal industry is one of little or no growth; whereas petroleum is expected to double its market during the same period. The magnitude of the competing groups, coal and oil, eclipses all other mineral and metal industries, whether they be measured in terms of tonnage or dollars. The petroleum group occupies a favored position so long as reserves of oil continue to be ample.

Emphasis on method of interpretation has been the prime idea in the preparation of this paper, but it is none the less realized that the forecasts open the door to economic and social problems of the first magnitude.

DISCUSSION

(*Howard N. Eavenson presiding*)

W. T. Thom, Jr.,* Princeton, N. J. (written discussion).—I have listened to this paper by Messrs. Hutchinson and Breitenstein with particular interest, having myself presented a paper on this general subject a year ago.[1] I question their forecasts of mineral fuel consumption in 1950, on the ground that it would be necessary to forecast the index of business activity for the country from now until 1950 in order to have any real basis for estimating what our fuel requirements will be as of that date. In the case of coal, we have already seen something of what increased efficiency of use can do to fuel consumption, even in a boom period. And if we are entered upon a period in which trimming fuel costs is an imperative measure of general economy, we will surely see material increases in the efficiency of use of both coal and petroleum, with the result that a growing energy demand may be derived from a stable or even a declining quantity of fuel.

Table 3 gives the impression of a direct relationship between total population and energy (and, indirectly, fuel requirements). Though this is doubtless the best assumption that can be made for purposes of estimation, it is not necessarily a correct one, for with essentially identical domestic populations, wide fluctuations in our fuel requirement have paralleled variations in our index of domestic industrial activity. Furthermore, with free outflow of manufactured exports our consuming population is to all intents and purposes greater than our national population, whereas with prohibitive tariffs against us this foreign increment of "effective" population is unavailable to us.

F. G. Tryon,† Washington, D. C.—The authors are to be complimented in having made the most painstaking analysis of past trends in fuel consumption that has yet been attempted. Others have traced past rates of growth without correcting for changes in the efficiency of fuel utilization, as the authors of this paper did. This is their unique contribution. It is to be hoped, however, that the authors will publish their assumptions as to thermal efficiencies past and present, both to enable others to verify their work and because data on this point are so fragmentary that any fresh material is of value. Very large assumptions are, of course, necessary. The efficiencies attained by certain of the larger consumers, such as the electric utilities and the railroads, are well known, but vast tonnages are consumed under the most conjectural conditions as to thermal efficiency. Who can say as to the present and past average efficiency of domestic heating? The Census of Manufactures shows the total quantity of fuel used in manufacturing, but nobody knows how much of it is used for generating power, how much for process steam, how much for direct heating applications. A large part of it goes simply for heating buildings: twenty-five million people have to be kept warm at their places of work, and the expenditure for this purpose is a large item in the country's energy budget. When the efficiencies attainable for these different purposes vary all the way from 15 to 80 per cent, who shall say what the average efficiency of utilization is? One lesson of the problem is the need of better records.

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* Associate Professor of Geology, Princeton University.

[1] W. T. Thom, Jr.: Relationships of Bituminous Coal to Oil and Gas. Third Int. Conf. on Bituminous Coal *Proc.* (1931) **1**, 82–114.

† Head Statistics Section, Coal Division, U. S. Bureau of Mines.

If the past consumption of tons or barrels of raw fuel be plotted, the curve rises sharply up to the end of the World War and then flattens. By correcting for thermal efficiency, the authors indicate that the growth in energy actually utilized has continued to increase since the war at much the old rate. If their assumptions are sound, the growth in per capita energy has been curiously regular. The future trend, however, remains uncertain, and the extrapolation of curves based on exponential rates of growth must be regarded with caution.

A. T. SHURICK,* New York, N. Y.—This paper is timely and it is gratifying to find the Institute alert to the pressing economic questions, repercussions from which have doubtless been experienced in greater or less degree by all members. This is a distinctive field that is the exclusive realm of the Institute, to which the business world is looking for authoritative data and guidance through these distressing times.

Banking and railroad interests, not to mention coal owners themselves, are and have been for some time, gravely concerned over the destiny of the coal industry. But serious investigation of the subject has been discouraged, largely by astonishingly optimistic predictions of various purely superficial studies. An example of this is the curve of anticipated future coal production that appeared in the TRANSACTIONS of seven years ago;[2] issued under the conservative imprint of the Institute, the banker has accepted this as authoritative, with the result that he is now convinced of the abortive results of real intensive research in determining where the industry is headed.

Only those who have had occasion to make an exhaustive investigation of this subject can appreciate the fallacy of the various superficial predictions advanced from time to time, which are to be greatly deprecated for the reason indicated above. It is evident that the authors of this paper have penetrated deeper into the fundamental issues than anything that has come to my attention, though certain hypotheses set up do not conform to conclusions arrived at in a similar study I was engaged upon in an advisory capacity a few years ago. This study was made for the purpose of effecting a decision on certain pressing questions of policy involving large capital expenditures, one item alone being a prospective purchase of about a million dollars in additional coal lands. A fairly liberal appropriation was available for the work, which required the services of four specialists some three and a half months, and it is interesting to know that the major conclusions of this study have been substantiated.

The authors' fixation on their subject title, coal and petroleum, has, I believe, distorted their perspective of the *tout ensemble*. Even at best we are still only fumbling in the dark with the vast and interlocking complexities of this subject, so that differences of opinion are inevitable at this stage; but I am currently of the opinion that the outstanding trouble with the coal industry will ultimately be localized more or less exclusively on the price factor. This phase was the dominant theme in the brief of the National Coal Association to the National Transportation Committee, wherein it was shown that the high freight rates of recent years had caused a loss of 200,000,000 tons of coal business a year to the carriers. Moreover, it is the pivotal issue before the Board of Reference in the present anthracite wage negotiations, the operators claiming that lower wages, with lower costs to the consumer, are an imperative prerequisite to salvaging the industry.

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* Consulting Mining Engineer.

[2] *Trans.* A.I.M.E. (1926) **74**, 431.

We developed this same hypothesis by bringing together the average spot price and production curves of bituminous coal, as shown in Fig. 2. The entirely unprecedented advance in the weighted average price of coal from $1.25 to $2.60 late in 1916, as shown in this graph, increased the 1917 coal bill of consumers from $672,000,000 to $1,388,000,000, some of the leading increases being: railroads from $192,000,000 to $340,000,000; industrial consumers from $220,000,000 to $458,000,000; domestic consumers from $71,000,000 to $148,000,000, etc. Theorize as we may on many of the abstract phases of this puzzling question in general, we are dealing here with cold, elementary facts of the first importance on the destiny of coal.

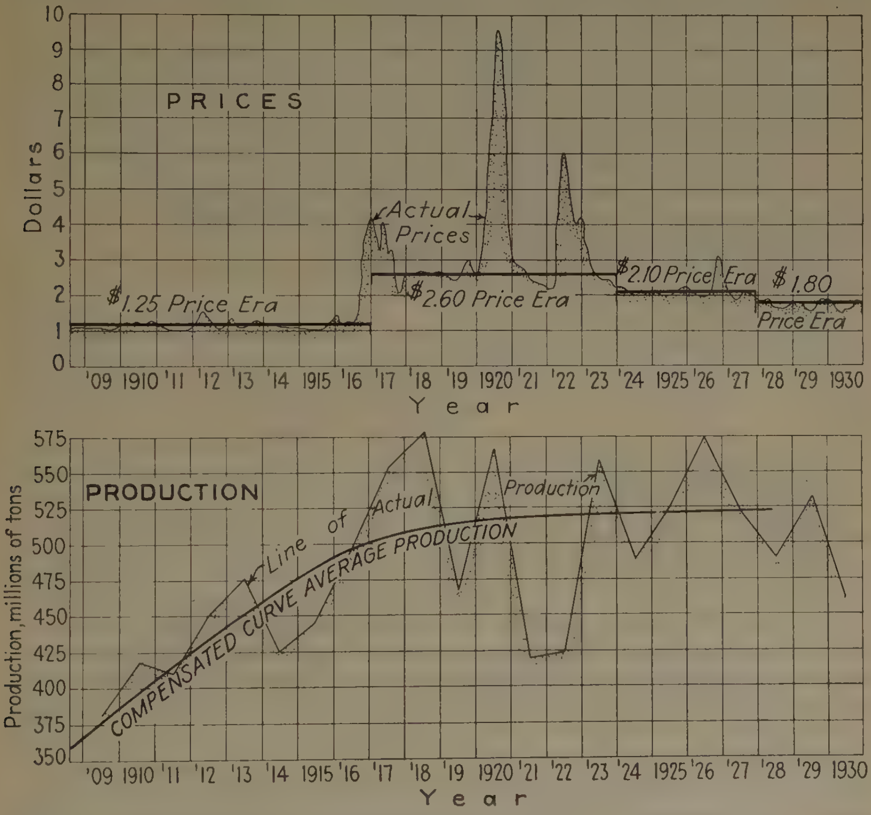

FIG. 2.—COMPENSATED CURVE AVERAGE PRODUCTION OF BITUMINOUS COAL WITH CONTRIBUTING FACTORS.

It was not to be expected that the alert American business executives would passively accept an additional burden of three-quarters of a billion dollars a year in their annual fuel bill. And we have their reaction written in unmistakable terms of a reduction in unit consumption by electric public utilities from 3 lb. of coal per kilowatt-hour in 1920 to 1.55 lb. in 1931; and a like reduction in railroad consumption from 174 lb. per gross ton mile to 118 lb. over the same period. The $148,000,000 increase in the annual fuel bill of the railroads as previously noted was the economic justification for the development of the modern costly superlocomotive using but 2½ lb. of coal per indicated horsepower at the drawbar, as compared with a general average of 5 lb. This is only typical of the sweeping economies that have been effected in all lines.

This hypothesis is clearly demonstrated in Fig. 2. On the lower left we have the compensated curve of average bituminous production ascending at a rate that has prevailed for around 100 years, until it is bent sharply over to the horizontal in the zone coinciding with the increase in the price of coal as shown above from $1.25 to $2.60. It seems clear that this doubling in the price of coal, or better, must take position as the outstanding decisive factor in the fate that has overtaken both the anthracite and bituminous branches of the industry. This not only precipitated the sweeping economies in fuel consumption previously noted, but also opened the door to the substitute fuels. Even though this high-price era of coal was contemporary with an era of vast overproduction in petroleum, it is probable that the price of coal was the decisive influence in the spectacular expansion in the development of the oil burner over the past decade; and the high price of coal likewise doubtless figured prominently in determining the economic justification of the great network of long-distance transmission lines that are now pouring floods of cheap natural gas into the larger consuming centers.

This subject has been developed in greater detail in the following studies by the writer:

Coal Prices Can Meet All Competition. *Forbes* (April, 1930). Reprinted in *The Black Diamond* (March 29, 1930).

Coal—A Pauper Baron. *The Magazine of Wall Street* (Sept. 3, 1932). Edit. rev. in *Min. & Met.* (December, 1932) and abstracted in *Saward's Journal* (Sept. 10, 1932).

Coal's Prospects Under the NRA Code. *Min. & Met.* (October, 1933).

An Estimate of the Probable Shift in Railroad Coal Traffic Under the Code. *The Annalist* (Feb. 2, 1933).

Chesapeake & Ohio Coal Traffic Prospects. Railroad Analyses, Inc.

Shift of Coal Tonnage to Northern Roads Follows Annalist Forecast of February 2. *The Annalist* (Aug. 3, 1934).

A. H. WILLETT,* Washington, D. C.—In commenting on Mr. Shurick's discussion, I wish to emphasize the severe competition that the coal industry faces from rival fuels and greater efficiency of combustion. The price of coal is highly significant, of course. It is likely that much of the high-efficiency apparatus already installed would not have been installed if coal prices had been as low as those now prevailing. A great deal of such equipment is not paying carrying charges, at present coal prices. However, past high prices brought out the high-efficiency installations that must now be dealt with. Present low mine prices for coal should be reflected in increased demand, but it is delivered price that really counts. The National Coal Association is trying to bring home the fact that the only place for further economies in delivered price is in a reduction in railroad freight rates on bituminous coal. Transportation now takes on the average over 60 per cent of the consumer's dollar. It is hard to estimate the coal tonnage that has been lost because of these high freight rates. When the operator cuts his price by 25 per cent the consumer gets only a 15 per cent reduction. It is difficult to meet competition under such circumstances.

In a certain sense there will always be overcapacity. If the country demands 13,000,000 tons of coal a week in the winter and 7,000,000 tons a week in the summer, nearly 100 per cent overcapacity must be maintained in the summer months. It is one of the aims of Appalachian Coals, Inc., the cooperative selling agency recently approved by the U.S. Supreme Court, to find a way for a group of operators to run part time and escape the financial loss involved in the present system of cut-throat competition.

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* Director, Bureau of Coal Economics, National Coal Association.

R. H. **Sweetser**,* New York, N. Y. (written discussion).—It was my good fortune to read the entire thesis by August J. Breitenstein upon which this condensed paper of Messrs. Hutchinson and Breitenstein is based. The original voluminous paper was sent to me for reviewing; at first it seemed an imposition, but as I read the paper the more interesting it became and the more evident was the fact that here was a new subject of the greatest importance presented in a most painstaking and convincing manner. The substantiating data assembled were carefully tabulated and certified, and authorities were recorded in this original paper, but in some cases it was not possible for the author to get facts bearing on the tonnages of anthracite and bituminous coal produced in certain periods. I refer especially to the lack of information regarding the percentages of ash in all classes of "hard" and "soft" coal during the years that production and consumption tonnages have been recorded.

Adjustments for "thermal efficiency" were made by Mr. Breitenstein in reference to the *use* of coal, but not for the ash content. It was not possible to determine accurately the real fuel value of the anthracite and bituminous coal tonnages produced during the decade ending with the year 1920, yet that period has been taken as the zenith for actual tonnage production. The aggravated (this word used advisedly) peak tonnages of coal produced during the war period contained millions of tons of earthy material recorded in the government statistics as "coal," but really it was worse than useless as fuel. The present-day tonnages of coal production expressed in quality-tons of that period would be increased by at least 10 per cent. The same thing happened again in 1920 and in 1923.

Just how far any adjustments for the percentages of ash would affect the peak-year tonnages of anthracite and bituminous coals, it is now impossible to determine, but it is probable that the curve for "energy from coal" shown in Fig. 1 would not have flattened out so much beginning with the year 1920, and that the per capita consumption of coal in 1917 would have been much less than 6.08 tons. The insistent demand from all classes of consumers for clean coal has resulted in the shipping of more efficient fuel in this present decade than ever before. Coal operators in general have recognized the fact that the keen competition from the gaseous and liquid petroleum fuels is due not only to convenience in handling but also to the absence of ash; therefore they are eliminating more and more of the extraneous ash in their products before sending them to market.

The value of such a paper as this lies in its substantial foundation, in great contrast to, and also antedating, the so-called "Technocracy." It illuminates the road that the coal industry must travel in order to merit the patronage of a market that is demanding low cost, convenience, cleanliness, and high efficiency in its fuel.

J. B. **Dilworth**,† Philadelphia, Pa. (written discussion).—One of the striking features of this interesting and timely paper is the results it presents of determinations of the "effective" energy derived from mineral fuels and water power in the United States for the past century. This must have involved a large amount of research and it is unfortunate that the present paper had to be so condensed that only a very brief summary of results could be included.

The uniform relation since 1900 of the total energy demand of the country to population is remarkable and encourages the belief that the rate of growth of this demand thus established may be used with considerable assurance in forecasting the country's energy demand for the next decade or so. When plotted as on Fig. 1 it gives an important "master curve" to guide the projection of individual curves of energy produced from different sources, such as coal, the petroleum group and hydropower.

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† Consulting Mining Engineer, Edward V. d'Invilliers Engineering Co.

To aid in analyzing the authors' forecasts of the proportion of total energy that will be furnished by bituminous coal and anthracite from the present time to 1950, Table 5 has been prepared from the figures in Tables 2 and 3.

TABLE 5.—*Proportion of Energy Furnished by Bituminous Coal and Anthracite*

Year	Coal Consumed, Thousand Tons	Effective Energy, Billions B.t.u.	Effective Energy, Billions B.t.u. per Thousand Tons Coal	
			Amount	Increase
1850	7,018	32,000	4.56	
1870	33,247	173,000	5.20	0.64
1890	156,382	955,000	6.11	0.91
1910	488,367	3,620,000	7.41	1.30
1930	517,019	5,443,000	10.53	3.12
1950	499,500	6,760,000	13.53	3.00

The last column in Table 5 shows the remarkable increase in efficiency with which coal has been used as a source of energy, particularly in the 20 years between 1910 and 1930. In this comparatively short time the effective energy obtained from 1000 tons of coal was raised 3.12 billion B.t.u., or 42 per cent, a greater increase than was secured in the preceding 60 years.

The authors' conclusion that in the next 20 years, or from 1930 to 1950, this remarkable performance will be repeated seems open to question. When it is recalled that the 20 years from 1910 to 1930 embraced a large part of the period when almost the entire United States was changing from small local steam or steam-electric power plants with low fuel efficiency to great central electric-generating stations of relatively high efficiency, and that such replacement is now almost completed, it seems doubtful whether the next 20 years will show any such improvement in utilization of coal energy as the past 20 years have afforded.

This has an important bearing on the probable consumption of coal. If, as seems likely, the 1930-1950 increase in efficiency amounts to but half the estimate here made, or 1.5 billion B.t.u. per 1000 tons of coal, the 1950 coal consumption would be increased some 56 million tons, or to a total of 556 million tons on the assumption here made of 6760 trillion B.t.u. of effective energy supplied by coal in that year.

Table 3 records the authors' belief that in the 20 years from 1930 to 1950 the enormous increase in the country's energy demand will be met largely by petroleum products and natural gas and that the rate of increase of energy from these sources that obtained from 1918 to 1930 will continue for another 20 years. As regards petroleum, automotive vehicles are the principal users of its products. Now passenger automobiles and trucks in the United States increased in number from about 6 million in 1918 to around 26 million in 1930, at which time there was one for every five people in the country. It seems impossible for this rate of increase to continue for 20 years more and therefore it is very doubtful whether the closely related 1918–1930 rate of increase in petroleum energy output will hold for that period, even allowing for the probable rapid future growth of the aeronautic industry.

Furthermore, if the demand for motor fuel *should* increase for the next 20 years at the rate of the last 20, the amount of petroleum utilized would probably increase at a much slower rate, owing to increasing proportions of motor fuel (mainly gasoline and kerosene) that will be recovered from crude oil. An evidence of this trend is that 100 gal. of petroleum afforded only 30 gal. of gasoline in 1920 as compared with about 50 gal. in 1930.

Stating the proposition generally, the source of energy that will be utilized most extensively in years to come will be the one that will give the most effective energy per unit of cost. On this basis it would seem that the increase in the country's energy demand over the next 20 years will be met more largely by coal than by petroleum and natural gas. It is probable that the slackening in the rate of increase in coal consumption in the past 20 years was due in no small degree to the relatively high price of coal during the war and post-war period. This justified large expenditures of capital to increase efficiency in steam power plants and invited competition from petroleum products. With the return of bituminous coal to pre-war prices, which is now an accomplished fact, there will be less incentive for increasing efficiency in its use as the law of diminishing returns will apply more and more effectively.

Moreover, the country's deposits of bituminous coal are so extensive and accessible to mining that, other things being equal, it will cost little or no more to produce a ton of coal in 1950 than it does today. This will hardly be true of petroleum. The average oil field, after reaching peak production, declines in output about 60 per cent the first year and 80 per cent by the end of the third year. Hence in normal times it is necessary to bring in a large number of new wells each year even to maintain current production. Generally speaking, new wells are deeper and more costly to drill than the ones they replace. Therefore it is highly improbable that either petroleum or natural gas will be produced as cheaply even 10 years hence as they are today.

Considering these factors in the fuel situation, it seems probable that by 1950 coal will be affording considerably more than the 46.6 per cent of the country's energy given in Table 1 and petroleum products correspondingly less than the 45.3 per cent there shown; also, that more than the estimated 499.5 million tons of coal will be required to furnish even the 6760 trillion B.t.u. of effective energy allotted to coal in Table 3. While no estimate of coal consumption even 10 years hence can be much more than a guess, there are indications that it will reach 800 million tons well before the middle of the century.

A. J. Breitenstein (written discussion).—In regard to Mr. Sweetser's question on the effect of correction for ash on our forecast: This question was brought up in particular reference to the poor quality of coal produced and sold during the World War period and the years 1920 and 1923. By the premises of our paper, the coal consumption figures are reduced to effective energy utilized by the application of estimates of thermal efficiency to statistics of consumption over the period of years considered in the study. These estimates of thermal efficiency were found to possess a rising trend from the earliest date considered up to the present time.

The improvement in the quality of coal during the past years has been a contributing factor to the decrease in the amount of coal required to do a unit of work, which is reflected in the rising trend in the estimates of thermal efficiency.

Since the forecasts are based upon the effective energy curve rather than the consumption curve, the effect of the increase in quality over a long period has been taken into account.

I do not believe that we should claim that our method has made a complete correction for excessive dirt in coal during the peak years mentioned, but I think that we can justly lay claim to a more reasonable forecast than had the actual peak consumption statistics been used as a base for forecasting.

Furthermore, any attempt to correct statistics for quality would, in a certain degree, invalidate the application of available data to estimates of increase in thermal efficiency. The factors of quality of coal and increase in thermal efficiency seem to be inextricably connected in their application to forecasting coal production as attempted in our paper.

In reading Mr. Dilworth's discussion, it strikes me that I did not make full use of, or at any time of my research fully realize, the value of population trends and forecasts made by the life insurance companies. I reviewed some literature by Louis I. Dublin, statistician of the Metropolitan Life Insurance Co., and obtained information on population trends which were applied in making the forecast. I did not base the forecast on an actual insurance company forecast.

Some investigations that I have made lately on population problems would indicate that my forecast of population in 1950 was too high. This, however, does not necessarily affect the accuracy of the energy demand curve because it is perfectly possible for the exponential rate of increase in energy demand to come into agreement with the exponential rate of increase in population at a later date than was indicated by the forecast.

W. S. Hutchinson (written discussion).—It seems appropriate to say that we are not unaware of the difficulties encountered in the collection of the data, in the assembling of the information, and in the contriving of its interpretation. Moreover, we are conscious of many unsolved questions of economic and social import beckoning to further research.

The forecasts were made in expression of a sense of professional duty, but no claim is made of a finality of numerical exactness. We do not discover, however, that anyone denies their reasonableness or the validity of the procedure adopted. Broadly speaking, we believe the method has withstood the test of critical discussion and it is our hope that the mineral fuels industries may find something useful in the results of our study.

We are impressed by the appreciative quality of the comment relative to our paper in the European journals: *Engineering*, *The Colliery Guardian* and *Petroleum Zeitschrift*, the latter having presented the paper in German and nearly in full. The following significant quotation is from *The Colliery Guardian:*

"Were a similar attempt made to forecast the position of the fuel industries in Great Britain, different methods would have to be employed and the results might not be parallel. There can be no doubt that the attempt is worth making, because economic and social problems of the first magnitude are involved."

I have not overlooked the comment made by others relative to effective energy derived from the mineral fuels; some raising the question whether it had been taken into account and others, I should say, assuming that it had not.

Mr. Breitenstein, in his thesis research for the Master of Science, Mineral Fuels: Supplies and Markets, devotes Chapter 4 to the subject of Effective Energy. This chapter is probably too long to be acceptable to A.I.M.E. for publication, but it may be of interest to set down in tabular form a statement of the subdivisions of the subject. The chapter comprises 49 pages of text, including 12 tables and 2 pages of bibliography of sources quoted. The divisions are as follows:

	Number of Pages
Introductory	2
Progress in Efficient Utilization of Fuel: 1700–1920; 1904–1929	5
Estimate of Thermal Efficiency in 1930	8
Bituminous Coal	3
Table XII: Estimate of Average Thermal Efficiency for Bituminous Coal in 1930	1
Table XIII: Effective Energy Derived from Bituminous Coal	1
Anthracite Coal	2
Table XIV: Estimate of Average Thermal Efficiency for Anthracite Coal in 1930	1

	Number of Pages
Table XV: Effective Energy Derived from Anthracite Coal	1
Petroleum and Petroleum Products	3
Table XVI: Average Thermal Efficiency 1930	1
Table XVII: Effective Energy Derived from Petroleum and Petroleum Products	1
Natural Gas	3
Table XVIII: Estimate of Average Thermal Efficiency for Natural Gas in 1930	1
Table XIX: Effective Energy Derived from Natural Gas	1
Natural Gas Gasoline	1
Table XX: Effective Energy Derived from Natural Gas Gasoline	1
Hydropower	7
Table XXI (Part 1): Effective Energy Derived from Hydropower, 1850 to 1930	1
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Discussion	2
Table XXII: Total Effective Energy Derived from all Mineral Fuels and Hydropower	1
Reference list	2

Analysis of Bituminous Coal Mines Suspended from 1923 to 1932, Inclusive

By Newell G. Alford,* Pittsburgh, Pa.

(New York Meeting, February, 1933)

The data in this paper were collected to show both the volume and speed with which bituminous mines have indefinitely suspended operation while economic advantage in the industry has so gravely declined. The study was limited to the bituminous coal-mining states east of the Mississippi River, wherein conditions affecting production and demand are more common respectively to all of the mining districts included and represent about 95 per cent of the nation's coal mining.

Data for the compilation of these statistics and for the construction of these graphs have been obtained in all cases from state mine department reports, except from those of Ohio and Virginia. In Virginia essential individual mine data were secured from an authentic source, but no individual Ohio mine information is obtainable, because all Ohio fact-gathering agencies are prohibited by law from revealing such data. In other states the facts were tabulated and later checked with the cooperation of Chief Mine Inspectors. In all, excluding Ohio, over 17,000 individual annual mine returns were examined. Special information was secured from each state mine department about suspensions in 1932.

Accurate information was not available from all states about mine abandonment due to exhaustion of coal reserves, and while these were completed for several states, it has not been shown because it was not perfected for all states. However, the total observed sample was comprehensive enough to show that exhaustion accounts for a very limited part of the suspended production.

Wherever mention is made of "suspended production" the largest annual output of the mine in the five years prior to suspension has been taken as the measure of its probable production had it continued to operate. A mine is not regarded in this category, as having suspended, when it merely ceases to produce. The word "suspension" is used only where there is a complete abandonment of activity for an indefinite period, as evidenced by the fact that both ventilation and pumping are discontinued.

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* Mining Engineer, Eavenson, Alford & Hicks.

The information has been arranged in many ways, and in this paper, for the most part, the figures have not been given, so the reader will have to rely for his information on the charts, from which he can obtain only their approximate values. However, further details will be supplied those desiring them.

Fig. 1 shows the annual suspended bituminous production of the area, plotted cumulatively; the annual production of that area and the potential capacity of its operating mines, the last being based on operation for 300 days per year. The production of the active mines and their capacity were supplied by the U. S. Bureau of Mines. In this graph the figures for Ohio are excluded.

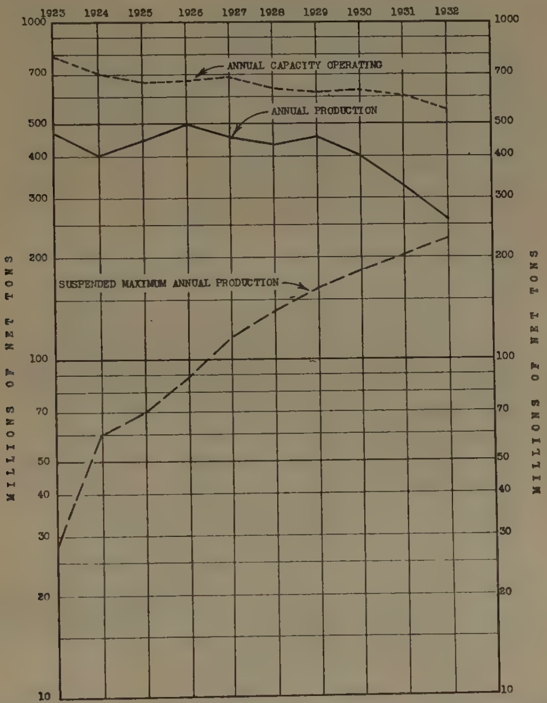

Fig. 1.—Bituminous coal production, potential capacity of operating mines and cumulative annual suspended production east of Mississippi River, excluding Ohio.

Fig. 2 shows, for the area, the number of bituminous mines that suspended in each of the several years. These are plotted both as totals and as segregated according to annual output. Figures for Ohio are excluded.

Fig. 3 presents the total annual suspended production of these same bituminous mines year by year. These are plotted both as totals and as segregated according to annual output, as in Fig. 2.

Fig. 4 shows the same total annual suspended production of these mines, and the number of suspending mines as such suspensions occurred year by year. Here again are totals and a segregation in groups, this time according to thickness, the seams over 42 in. thick being segregated from those of lesser thickness. Of the suspended production, 86 per cent was in coal over 42 in. thick and 67 per cent of the suspended mines worked the thicker seams. In these calculations Ohio and Virginia are both excluded.

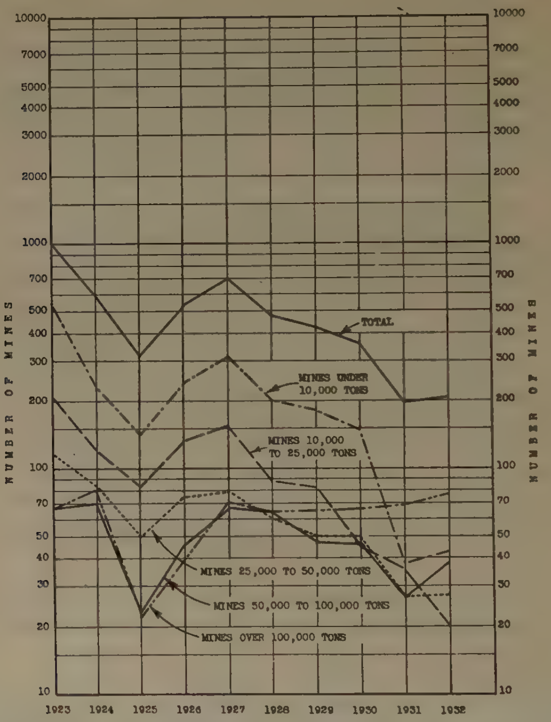

FIG. 2.—BITUMINOUS COAL MINES SUSPENDING, EAST OF MISSISSIPPI RIVER, EXCLUDING OHIO, CLASSIFIED BY ANNUAL MINE OUTPUT.

Fig. 5 groups the same mines so as to show the number of mines and production suspended in each year, north and south of the Ohio River.

Fig. 6, for the area east of the Mississippi, but in this case including Ohio, shows the total man-days worked annually in the bituminous coal mines, with the total annual man-days worked north and south of the Ohio River by years. (These figures are computed from data collected by the U. S. Bureau of Mines.)

Fig. 7 shows total annual production east of the Mississippi River, the total output for the United States and the annual production east of the Mississippi River produced by stripping, by underground mechanical loading and by ordinary underground hand labor.

Fig. 8 shows the number of men employed in all coal mines and in underground production and in strip pits, all east of the Mississippi River, with the total number of men displaced expressed cumulatively from 1923.

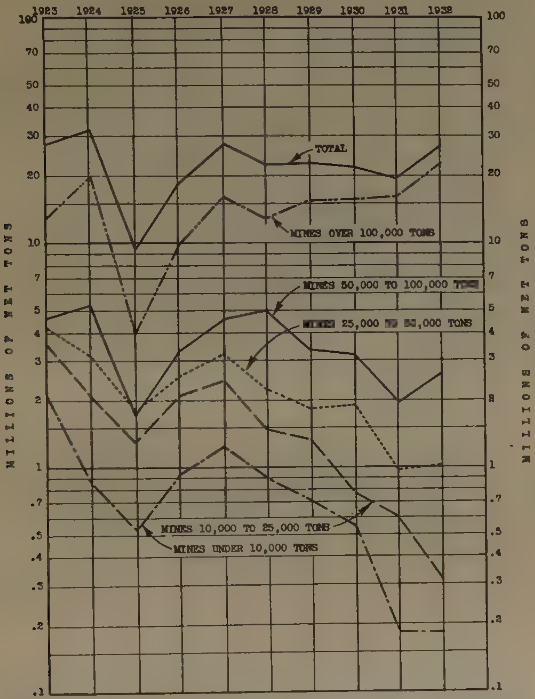

FIG. 3.—TONNAGE OF BITUMINOUS COAL MINES SUSPENDING IN AREA EAST OF MISSISSIPPI RIVER, EXCLUDING OHIO, GROUPED ACCORDING TO MINE OUTPUT.

Fig. 9 shows the significance of the output per man in the stripping operations, the total underground and strip-pit tonnage, combined and separately, as well as the tons per man-year for each classification. These figures, which are based on data obtained from the U. S. Bureau of Mines, include those from Ohio mines.

Fig. 10 shows the production, number of men employed and the number of mines working, and the cumulative number of mines shut down east of the Mississippi, excluding Ohio. Note the decline in the first three items.

Fig. 11 shows the number of mines operating, the cumulative total number of mines suspended, with graphs showing the number of strip pits operating and the number of underground mines using mechanical loading, all east of the Mississippi, excluding Ohio.

Fig. 12 shows the decrease in the total number of all bituminous mines operating east of the Mississippi, excluding Ohio, with their corresponding decline in production and in potential annual capacity, based on 300-day operation.

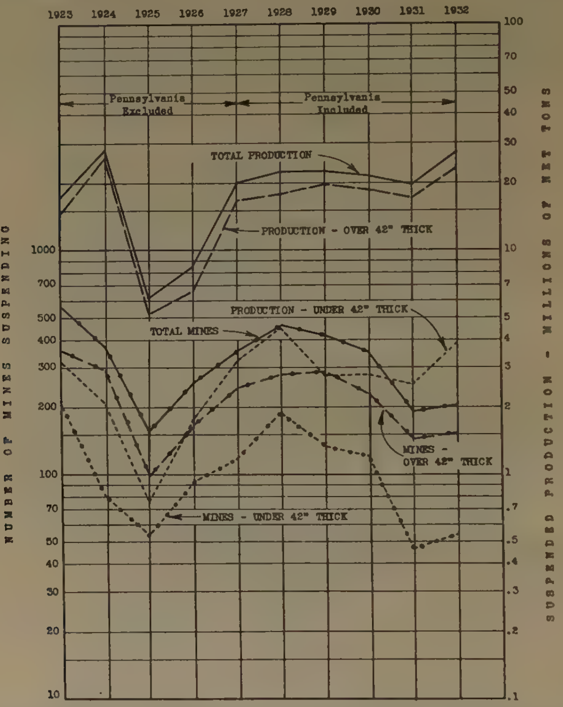

FIG. 4.—SUSPENDED BITUMINOUS COAL PRODUCTION AND BITUMINOUS COAL MINES SUSPENDING IN AREA EAST OF MISSISSIPPI RIVER, EXCLUDING OHIO, SEGREGATED ACCORDING TO SEAM THICKNESS.

DISPLACEMENT OF LABOR

The combined production of stripping and of mechanically loaded underground coal increased from 11.1 million tons in 1923 to 55.1 million tons in 1931, the mechanized coal advancing from 1.5 to 42 million tons in the same period. Since 1930, depression wages have reduced the differential that previously encouraged mechanized loading, and the previous rate of increase in this production sharply declined in 1930 and 1931; these figures are not yet available for 1932.

The annual tonnage of stripped coal east of the Mississippi increased through those eight years about 27.5 per cent to 13.15 million tons, although this volume was about 1 million tons higher in 1929 than in 1931 and almost that much in 1930. Meanwhile, this territory's mechanically loaded coal in 1931 was twenty-sixfold its 1923 production of 1.5 million tons. Despite this rapid rate of growth of coal production with mechan-

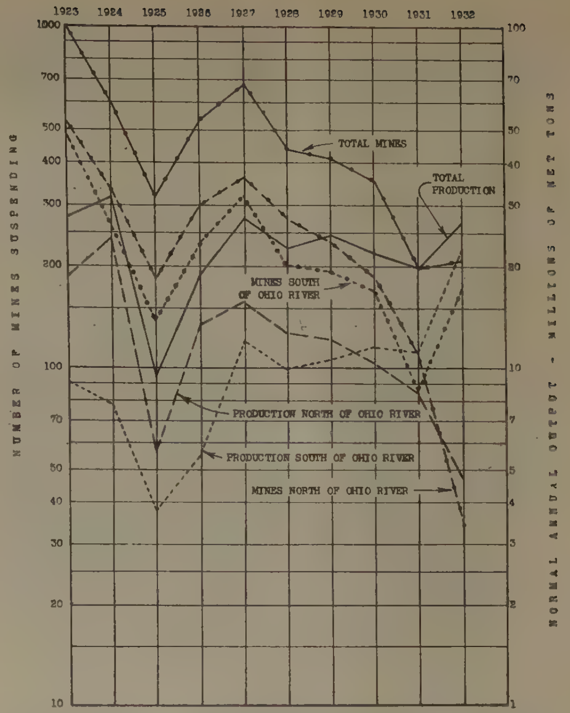

FIG. 5.—BITUMINOUS COAL MINES SUSPENDED AND THEIR TONNAGE IN AREA EAST OF MISSISSIPPI RIVER, EXCLUDING OHIO, SEGREGATED BY THEIR RELATIVE POSITION TO OHIO RIVER.

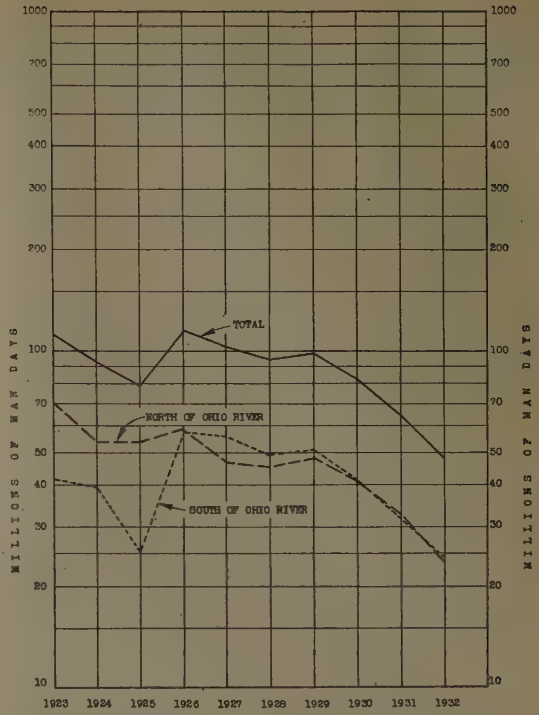

FIG. 6.—MAN-DAYS WORKED IN BITUMINOUS COAL MINES, EAST OF MISSISSIPPI RIVER, INCLUDING OHIO, WITH SEPARATE GRAPHS FOR THOSE LOCATED NORTH AND SOUTH OF OHIO RIVER.

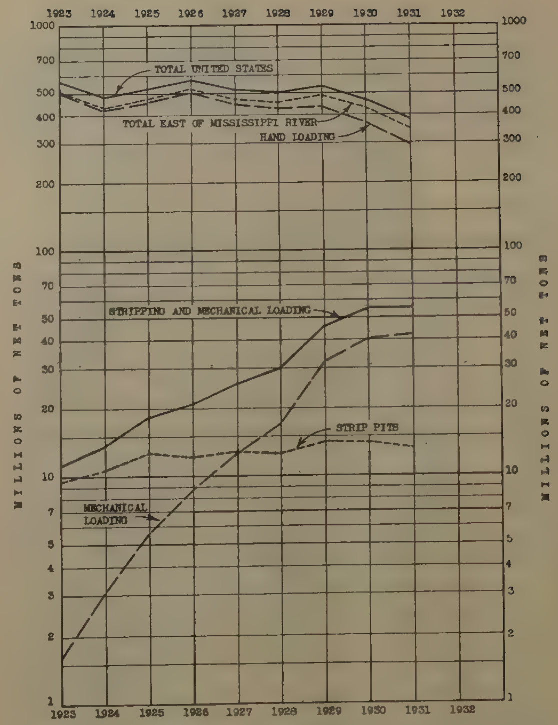

FIG. 7.—ANNUAL PRODUCTION OF STRIP-PIT, MECHANICAL LOADING AND HAND-LOADING BITUMINOUS COAL MINES EAST OF MISSISSIPPI RIVER, WITH TOTAL BITUMINOUS COAL PRODUCTION OF ENTIRE UNITED STATES.

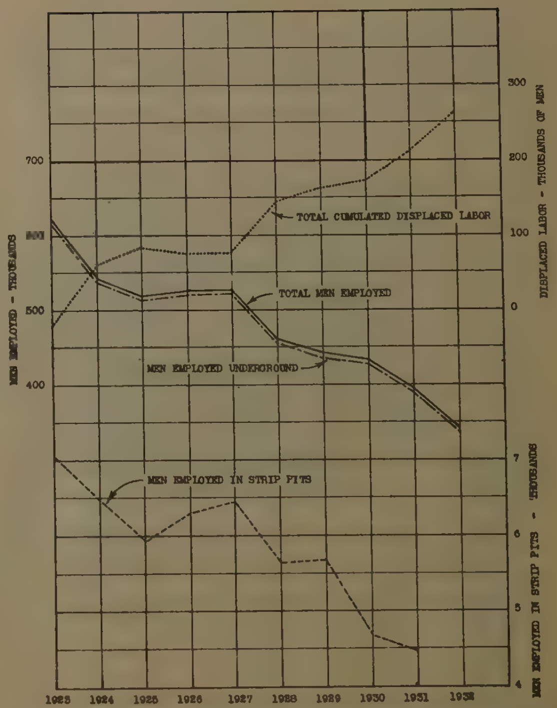

FIG. 8.—MEN EMPLOYED IN BITUMINOUS COAL STRIP PITS AND UNDERGROUND MINES EAST OF MISSISSIPPI RIVER, WITH TOTAL MEN SO EMPLOYED AND TOTAL CUMULATIVE DISPLACED LABOR.

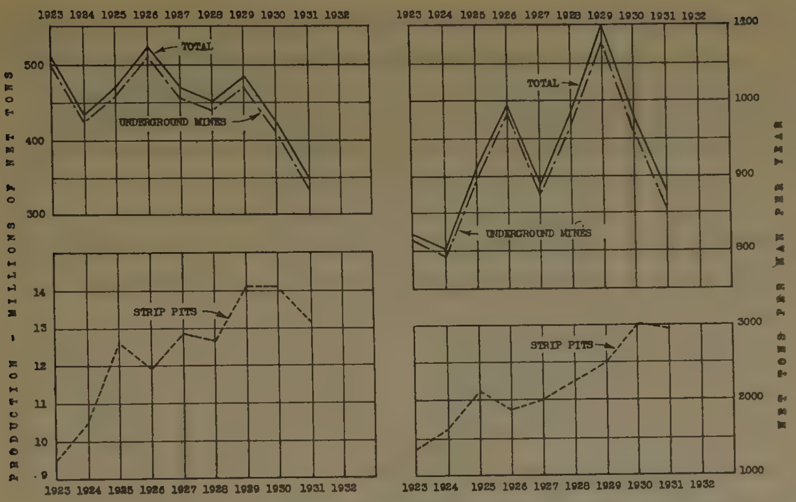

FIG. 9.—BITUMINOUS COAL PRODUCTION AND TONS PER MAN-YEAR FOR STRIP AND UNDERGROUND MINES, EAST OF MISSISSIPPI RIVER, INCLUDING OHIO.

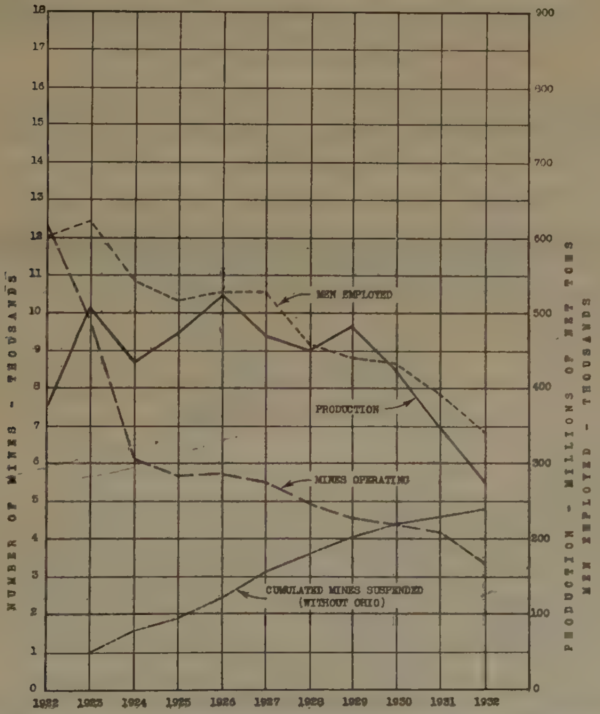

FIG. 10.—BITUMINOUS COAL PRODUCTION, NUMBER OF MINES WORKING AND TOTAL MEN EMPLOYED EAST OF MISSISSIPPI RIVER, EXCLUDING OHIO.

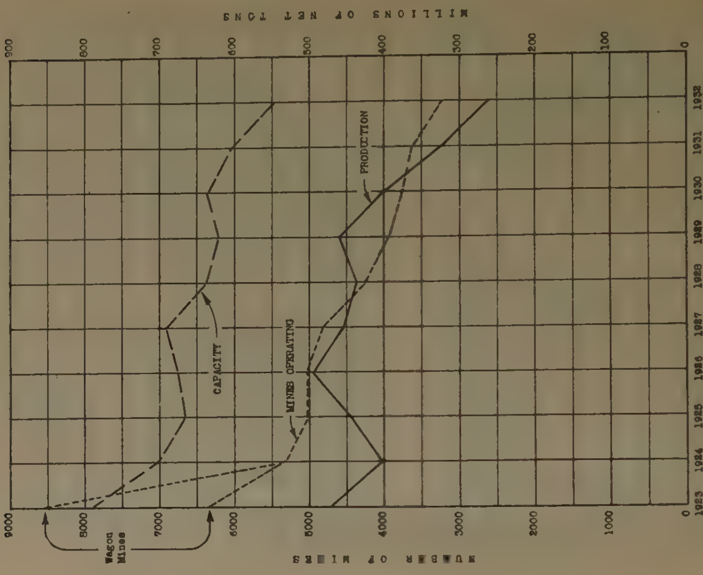

FIG. 12.—BITUMINOUS COAL PRODUCTION, TOTAL MINE CAPACITY AND NUMBER OF MINES OPERATING, EAST OF MISSISSIPPI RIVER, EXCLUDING OHIO.

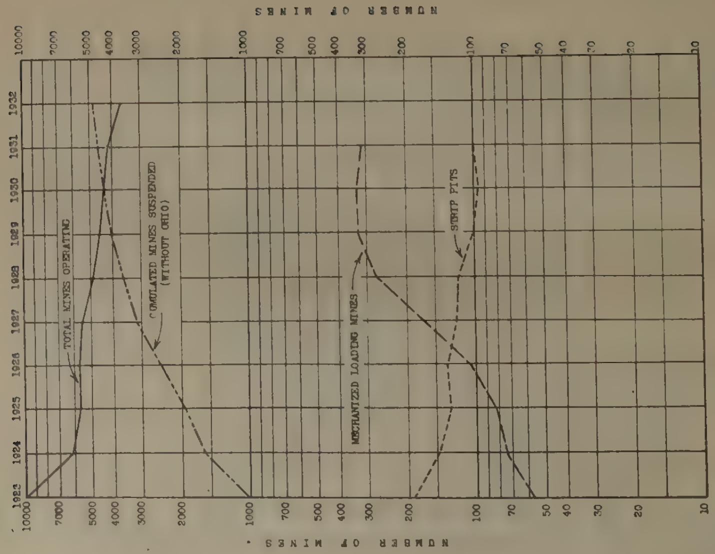

FIG. 11.—BITUMINOUS COAL MINES OPERATING AND SUSPENDED, MECHANIZED MINES AND STRIP PITS EAST OF MISSISSIPPI RIVER, EXCLUDING OHIO.

ical energy, only 12 per cent of 1931 output east of the Mississippi came from mechanized loading and 3.8 per cent from stripping.

Mechanically loaded production cannot be expected to resume its upward climb until wage rates materially increase in the nonunion fields and union rates are comprehensive enough to economically suit the wide range of conditions in these latter districts.

At the end of 1932 approximately 281,000 fewer men were employed in the industry east of the Mississippi than in 1923, of which about 50,300 had been displaced by mechanized loading, and approximately 6,200 by stripping operations. Of the remaining 224,500 men displaced, about 75,000 men have been removed by the collective effect of improved main haulage and gathering, increased use of track cutting equipment and better management with concentration of operations.

In practically every case, mechanical cleaning has been installed as an aid to the sale of coal, and while probably it has no direct bearing on the reduction of labor at the face, it seems certain that it has increased the productivity of face labor. While there has been no appreciable reduction in use of face labor, the net result has been a slight decrease in man-hours per ton at the working face.

Market shrinkage has displaced about 149,500 men from the industry through the past 10 years.

ACKNOWLEDGMENT

Cooperation is acknowledged from W. F. McKenney, Statistician, Koppers Coal Co.; J. J. Rutledge, John F. Daniel, W. B. Hillhouse, A. W. Evans, J. G. Millhouse, R. M. Lambie and A. C. Dally, chief mine inspectors for their respective states; Walter H. Glasgow, Pennsylvania Secretary of Mines; C. B. Neal, Secretary, Virginia Coal Operators' Association; F. G. Tryon and F. E. Berquist, of the U. S. Bureau of Mines; Byron M. Bird, Battelle Memorial Institute; Ezra Van Horn and others.

DISCUSSION

(*Eli T. Conner presiding*)

F. G. TRYON, Washington, D. C. (written discussion).—Mr. Alford has made a most valuable addition to our knowledge of the coal market. There is no point of greater concern to the bituminous operator than the extent to which the surplus of mine capacity has been liquidated by the 10 years of depression following 1923. We have all known that mines were shutting down in every field. Mr. Alford's study shows how many and where. Taking the fields east of the Mississippi (except Ohio), he finds the total number of mines suspending from 1923 to 1932 to be 4802. The total production of these mines when formerly in operation was 229,000,000 tons.[1]

---

[1] Alford evidently includes many small operations that would be classed by the U. S. Bureau of Mines as wagon mines, so that the figure of 4802 is not comparable with the Bureau's record of commercial mines. Inclusion of the wagon mines, however, has little effect on the tonnage.

How many of these mines should be considered potential producers and how many are permanently withdrawn from the productive capacity of the industry? The question cannot be answered accurately. The owners themselves do not know. The suspended properties are in all stages of collapse and disrepair, and the possibility of reopening depends in each case on physical conditions and most of all on the future of the market. Some of the closed mines were worked out and exhausted. It is known that the normal retirements through old age in the period before the war averaged about 200 mines a year, yet Alford concludes that "exhaustion accounts for a very limited part of the suspended production."

To throw light on the problem the Bureau of Mines has reviewed the statistical reports courteously supplied by operators during the past 10 years. The work was done under a cooperative agreement with the industrial research department of the University of Pennsylvania. Every year the Bureau asks for a report on the production of every known mine of commercial size. Where no coal was produced the operator is advised, "If idle, abandoned, or worked out please so state." If the owner reports that the mine is "abandoned" or "worked out" it is taken off the list; or if after due inquiry through the local post office no owner of the property can be found and the railroad records show no tonnage shipped for years, the mine is taken off the list. If, however, the owner reports the mine as "idle" it is kept on the list as a potential producer. Undoubtedly many owners report a property as "idle" long after it has ceased to have value, out of a natural reluctance to admit the loss of their investment. But until the Bureau is advised to the contrary, statistically speaking the mine is not dead.

The lists were therefore examined to see how many mines were still being carried as idle but not definitely abandoned. It was found that at the end of 1930 there were 1355 such mines. Some of them had mined no coal since 1923, and others had been shutting down in each year since. By years the record was as shown in Table 1.

TABLE 1.—*Mines Listed as Idle*

Number of Mines	Idle Since	Capacity, Tons	Number of Mines	Idle Since	Capacity, Tons
219	1923	17,000,000	313	1927	42,000,000
118	1924	19,000,000	210	1928	27,000,000
83	1925	13,000,000	261	1929	30,000,000
151	1926	12,000,000			
			1,355	Total carried on books as idle but not abandoned at end of 1930	160,000,000

As a mine closing 10 years ago would be more likely to reach the stage of definite abandonment than one closing 3 years ago, it was natural that the largest number of these statistically idle but not abandoned mines should consist of properties closed toward the end of the period. The largest single number, 313 mines with a capacity of 42,000,000 tons, dropped out during or after the strike of 1927. Capacity of 27,000,000 tons had been idle since 1928, and capacity of 30,000,000 tons had been idle since 1929.

In all, the 1355 mines had a capacity at 308 days of 160,000,000 tons a year, equivalent to, roughly, 100,000,000 tons of normal production. These figures have since been increased by further suspensions, and we estimate that over 200,000,000 tons of capacity was shut down but not permanently abandoned at the end of 1932.

Although the classification of idle but not abandoned is far from precise, the figures indicate a large shut-down capacity which could conceivably resume operations if the price were attractive enough. If the country were plunged into another World War, with acute shortage of fuel, many would doubtless reopen. On the other hand, at the price levels of recent years they are clearly out of the running. They were high-cost operations when they were forced to close. Since then, rapid deterioration has further impaired their competitive standing, while in the meantime the surviving mines have increased an inherent advantage by further mechanization. One practical test of the response of the shut-down capacity to price was afforded by the price flurry in the fall of 1926. The 9-month strike of the British miners had created a vacuum in the sea-borne coal trade, and the unexpected demand from overseas was added to heavy purchases at home, as consumers built up their stocks to prepare for the end of the Jacksonville wage agreement. From $1.91 in July spot prices jumped to $3.19 in November, with sales along the seaboard at still higher figures. Yet while some mines reopened in 1926, many others closed, and the capacity in operation at the end of the year was no greater than at the beginning. The test indicated that more than a short-lived flurry of high prices is needed to revive suspended mines. On the other hand, there can be no doubt that a period of sustained high prices and profits would bring scores or hundreds of them back into production.

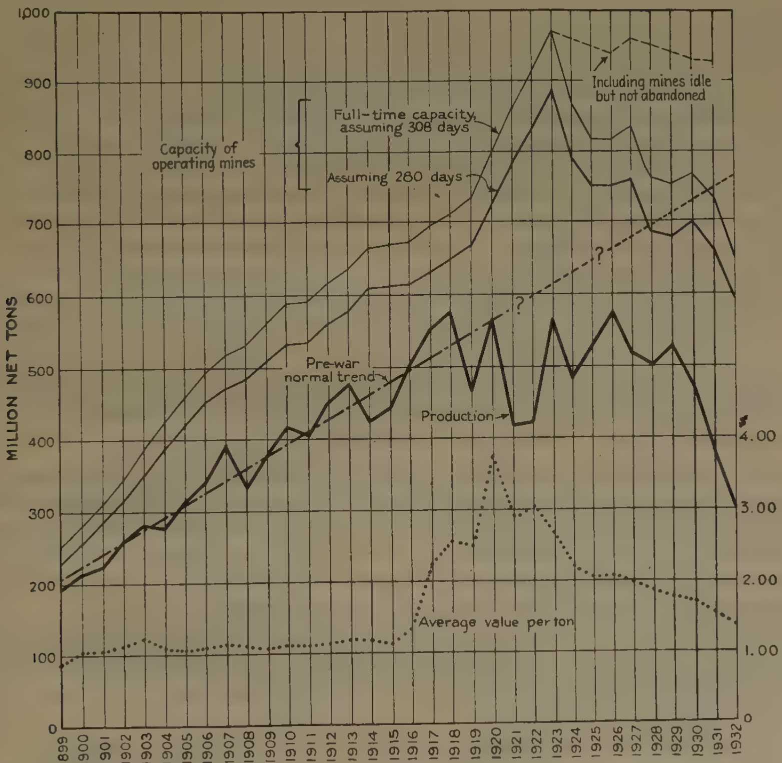

Fig. 13.—Trend of capacity of operating bituminous coal mines, 1890 to 1932, and of the total capacity including mines idle but not permanently abandoned, 1923 to 1932.

The trend of production and capacity is shown in Fig. 13. The capacity of the mines in operation has declined from 970,000,000 tons (308-day basis) in 1923 to 653,000,000 tons in 1932. Using the more conservative basis of 280 days a year, the capacity of the mines in operation in 1932 was 590,000,000 tons.

A. W. GAUGER,* State College, Pa. (written discussion).—Mr. Alford's analysis presents an interesting and graphic, albeit a sad, picture of the reaction of the bituminous coal industry to the irrevocable forces of economic change. The problem of too many mines and too many miners has faced this industry for at least 12 years and has carried with it social consequences that have undoubtedly contributed to the present situation.

As a result of the pressure for lower prices and lower costs, the industry has made astounding technologic advances resulting in the displacement of some 130,000 men. In addition to the social consequences attendant upon such a phenomenon, there is undoubtedly a certain measure of responsibility attached thereto. I do not suppose that it is possible to obtain any reliable data but it would be enlightening to know how many of the displaced miners were absorbed into other types of employment and how soon the change was accomplished. The time is perhaps near at hand when some provision will have to be made for labor that has been displaced by capital as a result of technologic advances. I do not favor or believe in the cessation of such advances, but certainly we can no longer escape the consequences thereof.

There is another result of the developments indicated by Mr. Alford's paper which is not mentioned directly by him; that is, the loss to posterity which must result in many cases from the abandonment of properties. Mr. Alford definitely states that exhaustion of reserves accounts for only a limited part of the suspended production. I should like to ask him if he could venture an opinion as to the amount of this reserve that is irretrievably lost because of the suspension.

N. G. ALFORD (written discussion).—In reply to Dr. Gauger's question about the amount of reserves that have been irretrievably lost through the operation of these suspensions, it is impossible to make such an estimate in the absence of the necessary information. Economic opportunity and mining ingenuity produced resumption of second mining in old pillars in Georges Creek district as well as in other old mining fields where it was once supposed that mines had been finally abandoned.

It is safe to assume that as the market for coal warrants the opening of abandoned mines they will again come into production but it is also probable that investors will be very critical of opportunities offered to them in such ventures, at least until after the current experience is dimmed by time.

As to the absorption of displaced miners by other industries, I know of no agency that has undertaken a comprehensive study of this movement, at least none of sufficient scope to answer the question for the area covered in this paper.

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* Director, Mineral Industries Research, Pennsylvania State College.

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